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MINE-VENTILATION STOPPINGS

WITH ESPECIAL REFERENCE TO COAL MINES IN ILLINOIS

BY

R. Y. WILLIAMS

ILLINOIS COAL-MINING INVESTIGATIONS COOPERATIVE AGREEMENT

[This report was prepared under a cooperative agreement with the Illinois State Geological Survey and the department of mining engineering of the University of Illinois]

WASHINGTON
GOVERNMENT PRINTING OFFICE
1915

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First edition. September, 1915.

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MINE-VENTILATION STOPPINGS.

By R. Y. Williams.

PURPOSE OF REPORT.

In this report an attempt is made to discuss the first cost of each of several types of mine-ventilation stoppings, as well as the total annual expense of maintaining a stopping during its required period of service. Such data furnish a basis for comparison of the efficiency of the different types, and are of great importance to an engineer planning the development of a mine. The data given were obtained through an investigation conducted under a cooperative agreement between the Bureau of Mines, the Illinois State Geological Survey, and the department of mining engineering of the University of Illinois. The report itself is published by the Bureau of Mines as a contribution to the literature on safety and efficiency in coal mining.

ACKNOWLEDGMENTS.

The writer wishes to thank the officials of the coal companies who contributed data and to express his appreciation of the assistance of the State mine inspectors and of the engineers of the Bureau of Mines and those working under the direction of the Illinois State Geological Survey and of the department of mining engineering of the University of Illinois. Special thanks are due George S. Rice, chief mining engineer of the bureau, for valuable suggestions on the compilation of data, to John T. Ryan, assistant mining engineer of the bureau, and S. O. Andros, mining engineer, for collecting much information on the efficiency of ventilation stoppings.

METHOD OF PRESENTING COST DATA.

The method adopted for presenting cost data in this report is as follows:

All the installation and maintenance charges, including first cost, maintenance, renewals, and emergency repairs, have been reduced to an annuity basis so as to show the amount of money that must be set aside annually to cover the upkeep and to provide for the amortization of capital at the current rate of interest for a given term of years.

Because the Illinois State mining law specifies the minimum amount of air that must be delivered to each man per minute, and because the sole purpose of each stopping built in crosscuts between the fan and the face is to prevent the air of the ventilating current from passing through the crosscut in which the stopping has been built, the cost of air leaks through stoppings may be found by determining the additional horsepower required by the fan to deliver air from the surface to the stopping through which the air leaks.

As an illustration of this method of presenting data and as an explanation of the terms used, let it be assumed that a number of main-entry stoppings are to be built for a service of 15 years. The items for labor and material in building the stoppings are included in "first cost"; the cost of labor and material required each year to make the stoppings efficient or tight is included under "maintenance charges." In case the material used in building the stoppings has an average life of only five years, the charge for the additional stoppings that must be built to give the 15-year service comes under the head of "renewal repairs." If the stoppings are of such a type as has been found in practice to be subject to destruction by shot firing, explosions, fires, water, etc., "emergency repairs" will be required. If air leaks through the stoppings and increases the volume of ventilation required of the fan, the amount of horsepower necessary to furnish the wasted air will determine the cost of stopping inefficiency, or the "cost of air leaks through stoppings."

TYPES OF STOPPINGS.

The word "stopping" as here used denotes a well or barrier built in a crosscut (termed "cut-through" or "break-through" in Pennsylvania) to prevent air from passing through. The word "brattice" denotes a partition built in a room or entry to conduct the air from the last crosscut to the face.

A great many types of stoppings are in use in the 840 coal mines in Illinois, the types varying from efficient concrete stoppings to inefficient stoppings constructed of powder cans and coal dust. The data in Tables 2 to 12 relate to the following nine types of stoppings:

1. Board stopping, rough lumber.
2. Board stopping, tongue and groove.
3. Board stopping, planed on two sides and painted.
4. Brick stopping, 4-inch wall.
5. Brick stopping, 8-inch wall.
6. Concrete monolith, 8-inch wall.
7. Concrete blocks, 8-inch wall.
8. Slate or gob, dry wall.
9. Slate or gob, faced with cement.

The list indicates the scope of the investigation herein reported. The main purpose is to suggest a way in which a superintendent or mine manager may solve mine-stopping problems by substituting particular values for those given in the tables.

SIZES OF STOPPINGS.

The dimensions of stoppings vary as follows:

Height	4 feet 6 inches to 9 feet.
Width	8 feet to 25 feet.
Thickness	1 inch to 20 feet.
Average surface area	79 square feet.

The variations in height are due to differences both in the thickness of the coal and in the physical properties of the beds. In some of the beds a strong top coal is left as the roof and in others a "draw slate" above the coal is removed in mining. Otherwise the stopping height is usually the same as the thickness of the coal bed.

The width of stoppings varies between the wide limits mentioned above because of the attempt at some mines to reduce the yardage in crosscuts by driving them full width. The objections to that method are that wide areas of roof are left unsupported and that wide stoppings increase the air leakage. Except in gob stoppings, the width of stopping is made 12 to 36 inches wider than the crosscut in order that the stopping may be "set in" a groove or channel which is cut in the rib to make a tight joint.

The thickness of the stopping varies with the material of which it is constructed, and, in the case of gob stoppings, with the amount of waste rock made in entry driving.

As crosscuts in Illinois mines should have an area of not less than 36 square feet when usedfor the passage of air, the smallest stopping will have a surface area, including the groove, of about 50 square feet. The higher limit should not be over 100 square feet. In the following tables, all data relating to stoppings are reduced to a common basis, that for a stopping having an area of 80 square feet, which is approximately the average size of Illinois stoppings.

NUMBER OF STOPPINGS REQUIRED.

In the usual room-and-pillar mine in Illinois, where the coal beds average 5½ feet in thickness and crosscuts are driven on 70-foot centers, the building of two entry stoppings for every acre of coal bed developed is required. In that State in 1913 there were 114 individual mines with an output of more than 200,000 tons of coal. This output required at each mine the development of about 50 acres and the construction of 100 new entry stoppings, of which 25 per cent were on main entries and 75 per cent on cross entries.

Table 1 shows the averages of many observations made to determine the total number of stoppings required for various acreages developed, and the total number of main and cross-entry stoppings in actual use at different periods in the development of a mine.

TABLE 1.—*Number of mine stoppings required for increasing mine development.*

Total mine development (acres).	Number of stoppings.			Number of stoppings in actual use.		
	Main entry.	Cross entry.	Total.	Main entry.	Cross entry.	Total.
100	50	150	200	50	150	200
200	100	300	400	100	300	400
300	150	450	600	150	400	550
400	200	600	800	200	400	600
500	250	750	1,000	250	400	650
600	300	900	1,200	300	400	700
700	350	1,050	1,400	350	400	750
800	400	1,200	1,600	400	400	800
900	450	1,350	1,800	450	400	850
1,000	500	1,500	2,000	500	400	900
1,500	750	2,250	3,000	600	400	1,000
2,000	1,000	3,000	4,000	800	400	1,200

COST OF ERECTING AND MAINTAINING STOPPINGS, ON A YEARLY BASIS.

FIRST COST.

Where a coal bed must be developed by expensive shafts it is considered necessary to have at least 2,000 acres tributary to each hoisting shaft. This acreage will require 4,000 stoppings, having a first cost of $20,000 to $60,000, a fact that shows how heavy an item in the development of a coal mine the cost of stoppings may be.

In the following table is given the first cost of nine types of stoppings. The items for materials and labor represent the average figures determined from a large amount of information submitted by coal operators. For each stopping, except the gob type, the labor item covering the building of the stopping includes the cost of cutting a latch or groove in the ribs for "setting in" the stopping.

TABLE 2.—*First cost of erecting nine types of stoppings, area of each assumed to be 80 (8 by 10) square feet.*

Kind of stopping.	Cost of materials.				Cost of labor.				Total first cost.
	Item.	Quantity.	Rate.	Amount.	Item.	Hours.	Rate.	Amount.	
Board, rough lumber.	Boards	0.10 board feet.	$25.00	$2.50	Delivery	2	$0.32	$0.64	
	Nails	1 pound	.04	.04	Building	10	.32	3.20	
	Posts	4	.08	.32	Calking	2	.32	.64	
	Total			2.86	Total			4.48	$7.34
Board, tongue and groove.	Boards	0.10 board feet.	35.00	3.50	Delivery	2	.32	.64	
	Nails	2 pounds	.04	.08	Building	10	.32	3.20	
	Posts	4	.08	.32	Calking	2	.32	.64	
	Total			3.90	Total			4.48	8.38
Board, s2s, painted.	Boards	0.10 board feet.	35.00	3.50	Delivery	2	.32	.64	
	Nails	2 pounds	.04	.08	Building	10	.32	3.20	
	Posts	4	.08	.32	Calking	2	.32	.64	
	Paint	1 can	.25	.25	Painting	2	.32	.64	
	Total			4.15	Total			5.12	9.27
Brick, 4-inch wall.	Brick	0.7 thousand	7.00	4.90	Delivery	2	.32	.64	
	Cement	1 barrel	2.00	2.00	Building	8	.32	2.56	
	Sand	0.4 yard	.42	.17					
	Total			7.07	Total			3.20	10.27
Brick, 8-inch wall.	Brick	1.2 thousand	7.00	8.40	Delivery	2	.32	.64	
	Cement	1.8 barrels	2.00	3.60	Building	14	.32	4.48	
	Sand	¾ yard	.42	.31					
	Total			12.31	Total			5.12	17.43
Concrete monolith, 8-inch wall.	Cement	1.2 barrels	2.00	2.40	Delivery	2	.32	.64	
	Sand	0.35 cubic yard.	.42	.15	Building	20	.32	6.40	
	Form, nails, tools, etc.			1.50					
	Stone	0.95 cubic yard	.50	.48					
	Total			4.53	Total			7.04	11.57
Concrete blocks, 8 by 8 by 16, 8-inch wall.	Blocks	90	.05	4.50	Delivery	2	.32	.64	
	Cement	2 sacks	.30	.60	Building	14	.32	4.48	
	Sand		.10	.10					
	Total			5.20	Total			5.12	10.32
Slate or gob, dry wall.					Building	22½	.32	7.20	7.20
Slate or gob, cement-faced, 1-inch.	Cement	0.7 barrel	2.00	1.40	Building	22½	.32	7.20	
	Sand	0.4 yard	.42	.17	Facing	4	.32	1.28	
	Total			1.57	Total			8.48	10.05

In order to show the annual expense necessary to bring about the amortization of the first cost at the end of the period of years during which service is required, the following formula may be used:

$$xr=\frac{Cr}{(1+r)^n-1}$$

Where C=First cost in dollars for construction.

n=Number of years of service required of stopping.

r=Current rate of interest.

x=Number of dollars that must be put away at r interest which will yield C dollars in n years.

and xr=Annual expense necessary to yield C dollars in n years.

With the aid of the formula, Table 3 following has been prepared to show the annual expense necessary to amortize first costs varying from $5 to $15 during periods of service of 3 to 20 years.

TABLE 3.—*Annual expense to amortize first cost of mine-ventilation stoppings.*[a]

First cost.	Length of service, years.									
	3	4	5	6	7	8	9	10	15	20
$5	$1.58	$1.14	$0.88	$0.71	$0.59	$0.50	$0.43	$0.38	$0.21	$0.14
$6	1.89	1.37	1.06	.86	.71	.61	.52	.46	.26	.16
$7	2.21	1.60	1.24	1.00	.83	.71	.61	.53	.30	.19
$8	2.53	1.84	1.41	1.14	.95	.81	.70	.61	.34	.22
$9	2.84	2.07	1.59	1.29	1.07	.91	.78	.69	.39	.24
$10	3.16	2.30	1.76	1.43	1.19	1.01	.87	.76	.43	.27
$11	3.47	2.53	1.94	1.57	1.31	1.11	.96	.84	.47	.30
$12	3.79	2.74	2.12	1.71	1.43	1.21	1.04	.92	.51	.33
$13	4.12	2.97	2.30	1.86	1.54	1.31	1.13	.99	.56	.35
$14	4.42	3.20	2.47	2.00	1.66	1.41	1.22	1.07	.60	.38
$15	4.74	3.43	2.65	2.14	1.78	1.61	1.30	1.15	.64	.41

[a] Figures calculated from formula $x = \frac{Cr}{(1+r)^n-1}$.

The annual sum to be charged to amortization of the first cost of the nine types of mine stoppings represented in Table 2 is given in Table 4 following:

TABLE 4.—*Annual expense to amortize first cost of nine types of stoppings.*

Kind of stopping.	Total first cost.	Annual sum to amortize first cost for period of 15 years.
Board, rough lumber	$7.34	$0.31
Board, tongue and groove	8.38	.36
Board, planed on two sides and painted	9.27	.40
Brick, 4-inch wall	10.27	.44
Brick, 8-inch wall	17.43	.74
Concrete monolith, 8-inch wall	11.57	.49
Concrete blocks, 8-inch wall	10.32	.44
Slate or gob, in dry wall	7.20	.31
Slate or gob, faced 1 inch with cement	10.05	.43

COST OF MAINTENANCE OF STOPPINGS.

The data presented in Table 1 indicate that most of the main-entry stoppings and many of those on the cross entries must be kept in good repair in order that the ventilation for any given development may be maintained in accordance with the State mining law. In carelessly constructed stoppings of lumber or gob material, it was frequently found that the maintenance cost amounted to $1 per stopping per year, a figure that is larger than the annual expense due to the first cost of a substantial stopping with amortization in a 15-year period. The annual maintenance charges, therefore, assume importance in mine economy.

The cost of maintenance is usually given as a yearly charge, and in that form may be combined directly with other cost items to obtain the total annual expense of stoppings. Maintenance charges on nine types of stoppings are given in Table 5.

TABLE 5.—*Annual maintenance charges for nine types of stoppings.*

[Stoppings 8 by 10—80 square feet; labor at 32 cents per hour.]

Kind of stopping.	Nature of repairs.	Frequency of repairs.	Cost of repairs per annum.		
			Material.	Labor.	Total.
Board, rough lumber	Lumber replacement	Once per annum	$0.50	$0.32	$0.82
Board, tongue and groove	Calking	do		.32	.32
Board, planed on two sides and painted.	Calking and painting	Once every 2 years	.20	.32	.52
Brick, 4 inches thick	Cementing on ribs	Once every 3 years	.10	.16	.26
Brick, 8 inches thick	do	Once every 4 years	.08	.11	.19
Concrete monolith, 8 inches thick.	do	do	.08	.11	.19
Concrete blocks, blocks 8 by 10 by 20 inches.	do	do	.08	.11	.19
Slate or gob in dry wall	Retamping at top	Once every 6 months		1.28	1.28
Slate or gob dry wall, faced with cement.	Refacing	Once per annum	.25	.64	.89

COST OF RENEWAL REPAIRS.

When the length of service required of a stopping is longer than the life of the materials used in construction, renewals will be necessary. The average main-entry stopping will have to give 15 years of service, and the cross-entry stopping 3 years. The choice of the type of stopping, therefore, should follow a comparison of the probable length of service of the stopping with the life of the materials to be used in its construction.

Table 6 gives data, obtained from mining companies in Illinois, on the life of the materials used in building stopping, and on the annual renewal expense necessary for a period of 15 years.

TABLE 6.—*Annual expense for renewal repairs on nine types of stoppings.*

[This table is of value in the consideration of main-entry stoppings.]

Kind of stopping.	Average life.	Cost of each renewal.	Annual renewal expense for period of 15 years.
	Years.		
Board, rough lumber	3	$7.34	$1.24
Board, tongue and groove	4	8.38	1.08
Board, s2s, painted	5	9.27	.80
Brick, 4 inches thick	8	10.27	.44
Brick, 8 inches thick	[a] 15		
Concrete monolith, 8 inches thick	[a] 15		
Concrete blocks, blocks 8 by 10 by 20 inches	[a] 15		
Slate or gob in dry wall	[a] 15		
Slate or gob dry wall, faced with cement	[a] 15		

[a] Or more.

COST OF EMERGENCY REPAIRS.

The foregoing discussion of renewals deals with the depreciation of stoppings through normal mining conditions. There are, however, extraordinary agencies that destroy stoppings, some of these agencies partly or wholly wrecking certain types of stoppings and yet in no way injuring other types. Such agencies are: Fires, windy shots, blown-out shots, explosions, local roof falls, general mine squeezes, and water.

In order to obtain the annual expense for emergency repairs on the different kinds of mine stoppings, information was obtained from coal operators regarding the number of stoppings and the cost of repairs each year because of these extraordinary agencies. The quotient obtained by dividing the total cost by the number of stoppings in use at the particular mine was charged to each stopping as the annual cost of emergency repairs. For example, at one mine the records showed that 18 stoppings were destroyed by windy or blown-out shots; the cost of each repair averaged $6.67; the total cost was $120.06; and, as 1,000 stoppings were in use, the expense to be distributed over each stopping was 12 cents per annum.

A compilation of the information obtained on the annual expense required for emergency repairs is given in Table 7 following.

TABLE 7.—*Annual expense for emergency repairs on different types of stoppings.*

Kind of stopping.	Number of stoppings destroyed per annum, per 1,000 in use, by—							Cost of each repair.	Annual expense per stopping for emergency repairs.
	Fires.	Windy shots.	Blown-out shots.	Explosions.	Local falls.	Squeezes.	Total.		
Board, rough lumber	2	5	5	2	2	2	18	$7.34	$0.13
Board, tongue and groove	2	5	5	2	2	2	18	8.38	.15
Board, planed on two sides and painted	2	5	5	2	2	2	18	9.27	.17
Brick, 4 inches thick	0	2	2	2	0	1	7	10.27	.07
Brick, 8 inches thick	0	0	0	2	0	0	2	17.43	.03
Concrete monolith, 8 inches thick	0	0	0	2	0	0	2	11.57	.02
Concrete blocks, blocks 8 by 10 by 20 inches	0	0	0	2	0	0	2	10.32	.02
Slate or gob, in dry wall	0	0	0	2	0	0	2	7.20	.01
Slate or gob dry wall, cement faced	0	0	0	2	0	0	2	10.05	.02

SUMMARY OF ANNUAL EXPENSE FOR ERECTION AND MAINTENANCE OF STOPPINGS.

A summary of the data in Tables 4 to 7 gives the total annual expense for erecting and maintaining each of nine types of stoppings during a period of 15 years.

TABLE 8.—*Summary of erection and maintenance costs of stoppings, reduced to the annual expense necessary for a period of 15 years.*

Kind of stopping.	Annual expense for 15 years.				
	First cost.	Maintenance.	Renewals.	Emergency repairs.	Total.
Board, rough lumber	$0.31	$0.82	$1.24	$0.13	$2.50
Board, tongue and groove	.36	.32	1.02	.15	1.85
Board, planed on two sides and painted	.40	.52	.80	.17	1.89
Brick, 4 inches thick	.44	.26	.44	.07	1.21
Brick, 8 inches thick	.74	.19	....	.03	.96
Concrete monolith, 8 inches thick	.49	.19	....	.02	.70
Concrete blocks, blocks 8 by 10 by 16 inches	.44	.19	....	.02	.65
Slate or gob, in dry wall	.31	1.28	....	.01	1.60
Slate or gob dry wall, faced with cement	.43	.89	....	.02	1.34

COST OF AIR LEAKS THROUGH STOPPINGS.

The sole purpose of a stopping is to prevent the air of the ventilating current from passing through the crosscut in which the stopping stands. The Illinois mining law specifies the minimum amount of air that must be delivered to each man per minute. With a given number of men, therefore, a definite quantity of air per minute should pass through the last crosscut of each split. If the stoppings are absolutely tight, the horsepower required by the fan to furnish this air may be obtained by the formula:

$$\text{Horsepower}=\frac{ksv^3}{33,000}$$

Where k=0.000,000,021,7 [a]

s=rubbing surface in square feet

v=velocity in feet per minute

If the stoppings are not tight, air will leak through each crosscut. The amount of this leakage will depend on the area of the hole or crack in the stopping and on the difference between the air pressures on the two sides. It is obvious that if air is lost before it reaches the working faces, additional air must be delivered by the fan in order to have the desired volume reach the last crosscut. Inasmuch as the horsepower required to move the air varies as the cube of the velocity, a serious leakage will require a great increase of horsepower to move the larger volume of air at the higher velocity. At Illinois coal mines a horsepower costs about $40 per year, including all engine and transmission losses. Therefore, if the quantity of air lost be known, it is possible to figure accurately the cost of the leaks through the mine stoppings. This cost of stopping inefficiency should be charged against the cost of stoppings quite as much as first cost, maintenance charges, etc.

[a] The value of k is empiric and is based on investigation by J. J. Atkinson. Other investigations have given different values. The friction constant should vary for the different points of air passage in the mine; but as the result would complicate without seriously affecting the present problem, the generally used value of k as given is assumed to operate continuously.

Table 9 gives results of observation made in Illinois mines to determine the general efficiencies of mine stoppings. This table shows for each mine the quantity of air delivered by the fan and the percentage of the total air reaching the last crosscut. The leakage is surprisingly large.

TABLE 9.—*Results of observation of actual efficiencies of mine stoppings.*

Mine No	County.	Fan.	Size of fan.	Quantity of air at fan.	Quantity of air at last crosscut.	Proportion of air reaching crosscut.
			Feet.	*Cubic feet.*	*Cubic feet.*	*Per cent.*
1	Franklin	Robinson	18 by 8	119,533	18,232	15.3
2	do	Paddlewheel		120,410	16,737	13.9
3	do	Stevens	14	65,300	6,160	9.4
4	do	Capell	20 by 7½	120,000	40,000	33.3
5	do	Paddlewheel	21 by 6	64,700	15,600	24.1
6	Williamson	do		59,827	7,035	11.7
7	do	Blakslee	20 by 6	96,000	22,140	23.0
8	do	Paddlewheel	21 by 6	50,400	3,840	7.6
9	do	Serocco	12	38,160	12,300	32.2
10	do	Paddlewheel	12	35,290	5,125	14.5
11	Macoupin	do		77,364	9,600	12.4
12	do	Miller	18 by 6	160,276	37,672	23.5
13	Christian	Paddlewheel	14 by 4	28,130	2,380	8.5
14	St. Clair	do	16 by 4	90,000	12,000	13.3
15	do	do	16 by 4	62,240	10,950	17.6
16	Sangamon	do	16 by 5	69,400	23,250	33.5

CONCRETE EXAMPLE OF LEAKAGE THROUGH STOPPINGS.

To show the effect of leakage of air through stoppings on the required horsepower at the fan to deliver a given volume of air at the last crosscut let the conditions outlined below be assumed. The conditions are chosen to represent, as nearly as possible, actual mining conditions in Illinois.

Area developed, acres	600
Size of air compartment of shaft, feet	6 by 10
Depth of shaft, feet	250
First split of air at foot of shaft; secondary splits 400 feet from shaft.	
Total number of splits	4
Height of entries, feet	6
Width of entries, feet	10
Number of men on each split	80
Total number of men in mine	320
Volume of air required at last crosscut on each split, cubic feet per minute	9,000
Total air in mine at last crosscut, cubic feet per minute	36,000
Number of main-entry stoppings on each split	75
Number of main-entry stoppings in mine	300
Number of entry stoppings in use on cross entries in each split	100
Number of cross-entry stoppings in use	400

In figuring the total horsepower on the air three factors must be considered, as follows: (*a*) All the air delivered by the fan passes through both the air shaft and the hoisting shaft; (*b*) one-half the total air travels each way from the foot of the shaft to the secondary

splittings; (c) one-quarter of the total air passes from the secondary splittings to each of the four splits. In the present problem it is assumed that no air is lost while the ventilation current is passing from the fan to the secondary splitting.

Table 10 shows the horsepower required on the air under varying ventilation efficiencies when the other conditions remained as assumed above. The term "ventilation efficiency" in this table represents the ratio between the volume of air reaching the last crosscuts and the volume delivered by the fan. In this particular problem it represents the efficiency of 4 lines of 75 main-entry stoppings in preventing the leakage of air through the crosscuts in which these stoppings are built.

TABLE 10.—*Horsepower required for different efficiencies of stoppings.*[a]

Section of mine.[b]	Percentage of air delivered by fan that reaches last crosscut.					
	100 per cent.	50 per cent.	40 per cent.	30 per cent.	20 per cent.	10 per cent.
Shaft (a=60; o=32; l=500):						
Quantity of air, cubic feet per minute	36,000	72,000	90,000	120,000	180,000	360,000
Velocity of air, feet per minute	600	1,200	1,500	2,000	3,000	6,000
Horsepower required	2.27	18.18	35.51	84.17	284.07	2,272.60
Shaftbottom to secondary splits (a=60; o=32; l=400):						
Quantity of air, cubic feet per minute	18,000	36,000	45,000	60,000	90,000	180,000
Velocity of air, feet per minute	300	600	750	1,000	1,500	3,000
Horsepower required	0.45	3.64	7.10	16.84	56.82	454.52
Four separate splits (a=60; o=32; l=24,500):						
Quantity of air, cubic feet per minute	9,000	18,000	22,500	30,000	45,000	90,000
Velocity of air, feet per minute	150	300	375	500	750	1,500
Horsepower required	6.96	27.84	43.50	77.33	174.00	695.99
Total horsepower required	9.68	49.66	86.11	178.34	514.89	3,423.11
Cost at $40 per horsepower	$387.20	$1,986.40	$3,444.40	$7,133.60	$20,595.60	$136,924.40

[a] Efficiency is considered 100 per cent when all the air from fan travels throughout the mine. When efficiency is less than 100 per cent all the air is assumed to travel through a distance equal to l multiplied by the figure for the percentage of air reaching the last crosscut.
[b] a—area; o—rubbing surface; l—length of entry.

DATA ON COST OF AIR LEAKS THROUGH STOPPINGS.

METHOD OF OBTAINING DATA.

In obtaining the data and in making the computations for determining the cost of horsepower lost by the leakage of air through mine stoppings the following method was used:

In the entry or part of an entry chosen the stoppings were all of the same type of construction and the travel of the air was confined to the entry. The stoppings were examined, and their number, type, cost, condition, etc., were recorded. The volumes of air passing the starting and the finishing points on the entry were carefully measured.

COMPUTATIONS MADE.

The difference between the volume at the starting point and that at the finishing point was taken to represent air lost through the stoppings. As all the stoppings were of the same type in each test and as their number was small, it was assumed that the air leaked equally through each stopping.

From a series of tests an average figure was obtained to represent the volume of air that under normal conditions would pass through each of the different types of stoppings. The volume varied widely for each type because of differences in air pressures and in tightness. The average values were used in figuring the stopping costs in the example cited above. It was assumed that one-half of all air "lost" in traveling through the mine was short-circuited through doors.

Table 11 shows the average volume of air lost through each stopping, the resulting total mine efficiency, and the annual cost of air leakage for each of nine types of stoppings.

TABLE 11.—*Cost of air leaks through stoppings of various types.*

Kind of stopping.	Ventilation efficiency of entire mine.	Air leakage per stopping per minute.	Annual cost of air leakage per stopping.
	Per cent.	*Cubic feet.*	
Board, rough lumber	15	340	$74.49
Board, tongue and groove	25	180	19.06
Board, planed on two sides and painted	35	171	8.04
Brick, 4 inches thick	70	26	1.48
Brick, 8 inches thick	85	10	.95
Concrete monolith, 8 inches thick	90	6	.83
Concrete blocks, blocks 8 by 10 by 20 inches	90	6	.83
Slate or gob dry wall	20	240	34.33
Slate or gob dry wall, faced with cement	35	171	8.04

SUMMARY OF COSTS OF ERECTION AND MAINTENANCE OF STOPPINGS.

Table 12 gives the total yearly expense for each of the nine types of stoppings. It should be remembered that the figures apply to mines in which 100 cubic feet of air per man per minute is delivered to the last crosscut on each main split, and where approximately 200,000 tons of coal is produced per annum. In smaller mines the cost of air leaks might be less, owing to lower ventilating pressures.

TABLE 12.—*Annual expense for each main-entry stopping having a length of service of 15 years, in mine with annual production of 200,000 tons.*

Kind of stopping.	First cost.	Cost of maintenance.	Cost of renewal repairs.	Cost of emergency repairs.	Cost of air leaks.	Total.
Board, rough lumber	$0.31	$0.82	$1.24	$0.13	$74.49	$76.99
Board, tongue and groove (well calked)	.36	.32	1.02	.14	19.06	20.91
Board, s2s, painted (well calked)	.40	.52	.80	.15	8.04	9.93
Brick, 4 inches thick	.44	.26	.44	.07	1.48	2.69
Brick, 8 inches thick	.74	.19	....	.03	.95	1.91
Concrete monolith, 8 inches thick	.49	.19	....	.02	.83	1.53
Concrete blocks, 8-inch wall	.44	.19	....	.02	.83	1.48
Slate or gob, in dry wall	.31	1.28	....	.01	34.33	35.93
Slate or gob, dry wall, faced with cement	.43	.89	....	.02	8.04	9.38

DESCRIPTIONS OF VARIOUS TYPES OF STOPPINGS.

GOB STOPPINGS.

Most of the stoppings in Illinois are built of gob material. The dimensions vary as follows: Height, 4½ to 7½ feet; length, 6 to 25 feet; thickness, 6 to 20 feet. Generally the gob material is merely thrown into the crosscut; frequently one or both sides are built up with a dry wall of "slate;" sometimes the wall is laid in cement or is luted with clay, and occasionally a concrete exterior is made by building a form about 12 inches in front of the wall and tamping concrete in the space thus obtained.

The following general objections apply to the use of gob material for stoppings:

1. The labor item in the first cost is high (see Table 13). Except where the hoisting equipment is overcrowded in handling the coal output, the gob material in excess of the storage room underground could be more cheaply handled by loading it into cars and dumping it into some form of conveyor on the surface by means of a deflecting bottom in the weigh pan.

2. The maintenance is usually expensive, costing from 60 to 90 cents per stopping per year.

3. In time of danger, the gob stopping is not tight enough to seal off fires, leaves much débris to be cleaned up after an explosion, is not packed tight enough to assist in preventing the spread of a mine squeeze, and usually contains much dry coal dust, which aids in the propagation of an explosion.

4. A large volume of air leaks through a gob stopping.

Plate I, *A*, shows a gob stopping 6 feet thick and faced on each side with a dry wall of "slate" and roof coal. Some stoppings of this type showed a leakage of air amounting to 1,300 cubic feet per minute per stopping.

In one mine, because of a serious leakage of air, each of the gob stoppings was coated with 12 inches of clay. After the clay had dried, it cracked badly and allowed the air to pass. The stopping was then plastered with wood-fiber cement as shown in Plate II, *A*, the result being decidedly satisfactory. At a total cost of $1.60 per 80 square feet of stopping, the air leakage was reduced to 28 feet per stopping.

Table 13, showing labor costs for building gob stoppings of different sizes, follows:

TABLE 13.—*Cost of labor for building gob stoppings of different sizes.*

Area.	Depth.	Cost of labor.		
		Per stopping.	Per square foot.	Per 80 square feet.
Square feet.	*Feet.*			
55	8	$7.59	$0.138	$11.04
90	5½	10.50	.117	9.36
132	9	14.25	.108	8.64
84	9	8.49	.101	8.08
60	....	5.29	.088	7.04
36	....	2.62	.073	5.82
143	6	9.25	.065	5.18
52	6	3.37	.065	5.18
147	6	7.86	.054	4.32
187½	12	13.10	.070	5.60
Average..	....	....	.088	7.04

BOARD STOPPINGS.

Board stoppings are widely used in Illinois coal mines. They may be easily and cheaply built and are suitable for the crosscuts in short butt entries and for the break-throughs in rooms. On main entries where the ventilating pressure causes a high water gage, the cost of air leaks in board stoppings is large; and at places where the period of service is over 3 years, the maintenance charges and the renewal repairs are costly.

Tongue-and-groove boards are often substituted for rough lumber in order to obtain tight stoppings, and may be given additional life by painting. The importance of tight joints is evident. Tongue-and-groove lumber and wood-fiber plaster are materials that can be used to build tight stoppings; but in usual practice stoppings made of smooth boards are leaky. There are two reasons for leakage.

In the first place, plaster will not adhere firmly to smooth boards that are subjected to the vibration incident to mining operations. Even if the plaster would adhere, little could be gained in plastering the surface of the stopping. Tongue-and-groove boards will not leak, and paint is a far better preservative than plaster. Wood-fiber plaster, however, is necessary at the joints where the stopping touches the ribs, roof, and floor, and at those joints a roughened surface must be offered in order that the plaster will hold. Roughening may be accomplished by scratching the surface of the stopping with a sharp pick; or better still by nailing a strip of lath, prop bark, or rough wood within 1 inch of the four edges of the stopping.

In the second place, many board stoppings leak because the attempt has been made to build them without cutting a V-shaped groove into the rib to hold the cement.

A. GOB STOPPING FACED WITH DRY WALL OF SLATE AND ROOF COAL.

B. CONCRETE-BLOCK STOPPING.

Figure 1 shows a method for making a tongue-and-groove stopping efficient as long as the wood lasts.

Plate II, *B*, shows a rough-lumber stopping coated with wood-fiber cement. The loss of air through such a stopping was 112 cubic feet per minute, being considerably less than through an uncoated stopping.

Figure 2 shows a rough-board stopping that differs somewhat from the usual type. The boards are nailed to the props like clapboards on a house except that they are upside down. The upper edge of each board is beveled and offers a recess which holds mortar much more securely than when plastered on the face of an ordinary stopping.

A cheap and efficient stopping for room entries may be constructed by nailing Berkytt lath to props and covering the lath on the intake side with a ½-inch coat of wood fiber. The Berkytt lath are boards,

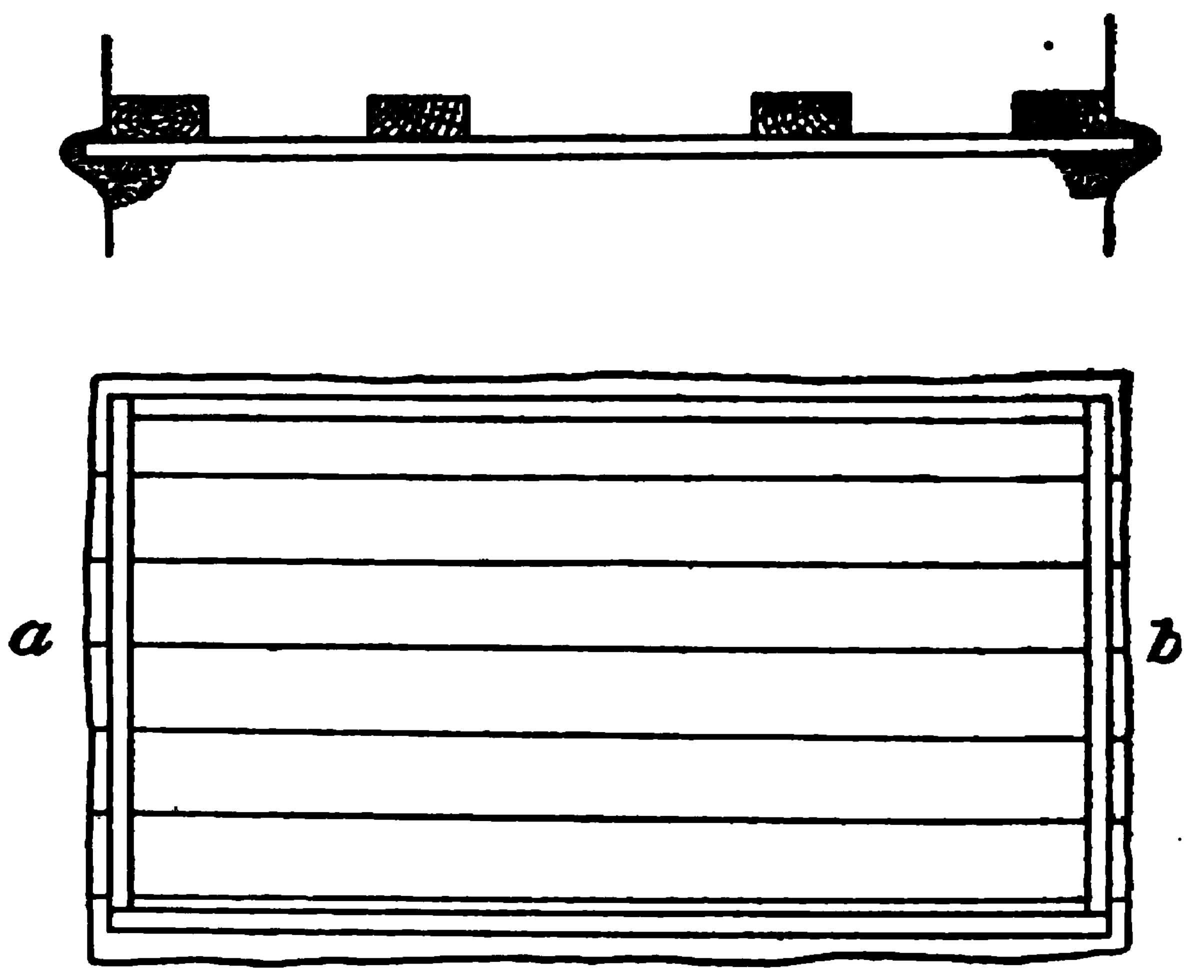

FIGURE 1.—Properly constructed stopping made of tongue-and-groove lumber. Note braces and grooves cut into ribs.

on one surface of which grooves are cut for the retention of the wood fiber.

The items that enter into the first cost of a stopping of this kind are reported as follows for a size 7½ by 9 feet:

Material:		
Wood fiber, 100 pounds	$0.40	
Byrkett lath, 70 square feet, at $11.50 per 1,000	.81	
Props, 3 at 8 cents	.24	
Nails	.05	
		$1.50
Labor:		
Setting props and nailing lath, 4 hours at 35½ cents	1.42	
Mixing wood fiber and plastering, 4 hours at 35½ cents	1.42	
		2.84
Total		4.34

In the figure for total cost, no allowance is made for cost of delivery of materials from the mine entrance to the crosscut in which the stopping is to be built, and no item is given for the cost of cutting the latches or grooves in the ribs. In order that the figure for total cost may be compared with those given in Table 2, the following additions should be made:

Total cost as above for area of 67½ square feet	$4.34
Addition for larger stopping, 80-square-foot area	.80
Addition for delivery of material	.64
Addition for cutting groove in ribs	.64
Total	6.42

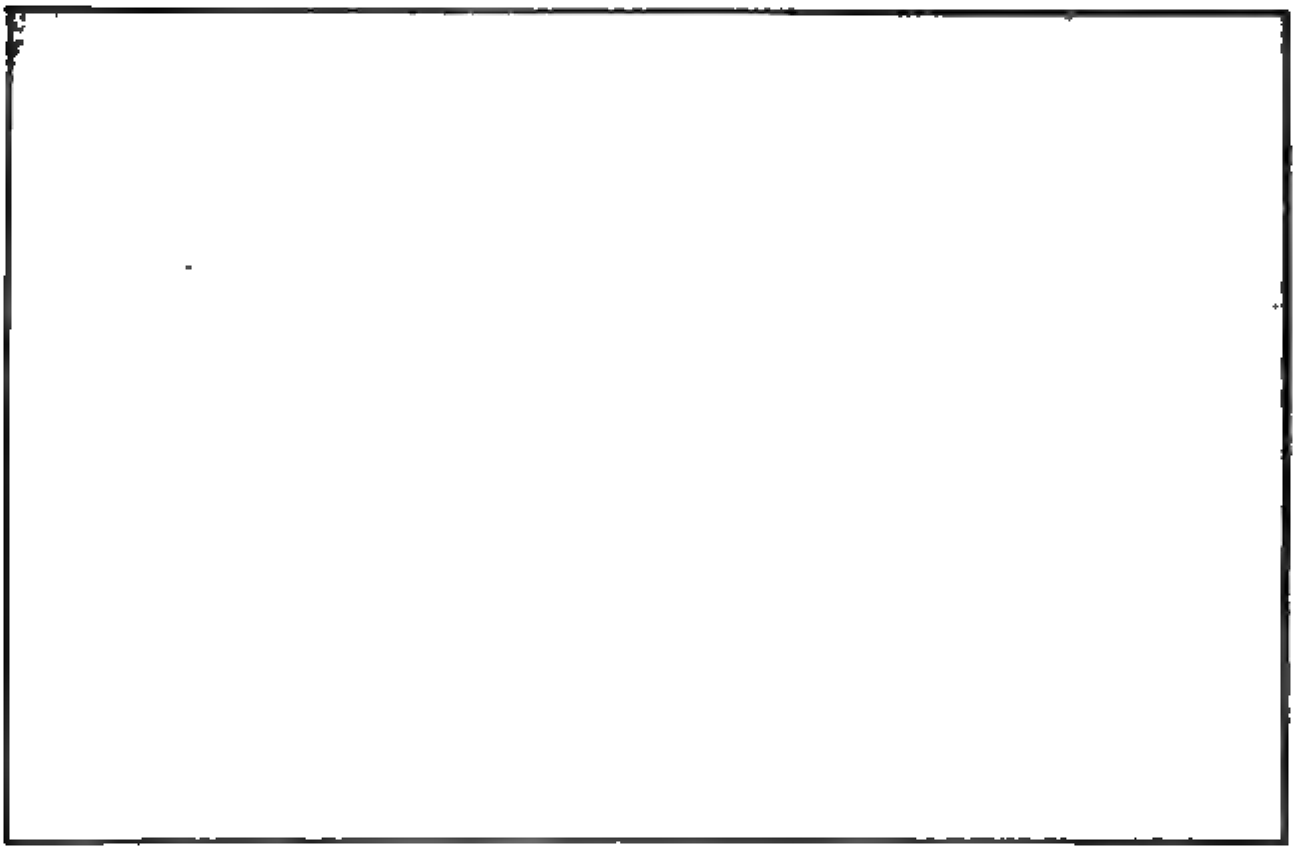

FIGURE 2.—Rough-board stopping, inverted clapboard type.

BRICK STOPPINGS.

In a few mines, bricks are used in building efficient stoppings. Operators report that except in locations where brickyards are close to the mines, it is cheaper to use concrete than brickwork for permanent stoppings.

At one mine where 8-inch brick stoppings were shaken down by the shooting, a remedy was suggested by George S. Rice, chief mining engineer of the Bureau of Mines. A double-arched stopping was built, each arch being 4 inches thick, and the two arches being opposed and touching at the center, as shown in figure 3. The space between the walls was filled with dirt and clay lightly tamped.

A. GOB STOPPING FACED WITH 12 INCHES OF CLAY. THE CRACKS IN THE CLAY ARE BEING PLASTERED WITH WOOD-FIBER CEMENT.

B. ROUGH-LUMBER STOPPING COATED WITH WOOD-FIBER CEMENT.

CONCRETE STOPPINGS.

In many mines, concrete is the most suitable material for use in the construction of efficient stoppings in the crosscuts between main air passageways. Concrete stoppings may be constructed at reasonable first cost, they have low maintenance charges, they are durable,

FIGURE 3.—Plan of arched brick stopping.

incombustible, and unaffected by water, and they have a high bearing strength to assist in supporting the roof.

The installation costs as reported by several operators are as follows:

TABLE 14.—*First costs of concrete-block stoppings.*

Size of blocks.			Cost of blocks.	Cost of cement mortar.	Cost of delivery into mine.	Cost of building stopping.	Total first cost.
Height.	Length.	Breadth.					
Inches.	*Inches.*	*Inches.*					
8	20	5	$3.96	$0.31	$0.71	$5.24	$10.22
12	24	8	8.36	.36	.71	7.25	16.68
[illegible]	20	6	2.16	.32	.71	5.46	8.65
8	18	6	6.57	.37	.71	2.84	10.49
8	16	8	4.10	.52	.71	2.62	7.95
8	[illegible]	8	3.28	.52	.71	2.84	7.35
							10.74
Average cost			4.74	.40	.71	4.38	10.29

At many mines, where care has been used in the construction of the joint at the roof and the ribs, there have been no maintenance charges. At one mine it was reported that this item amounted to 5 cents per annum per stopping, based on an expenditure of 50 cents per year for every 10 stoppings.

Renewal repairs are unnecessary in concrete stoppings because, under ordinary mining conditions, the life of a concrete stopping is longer than the period during which service is required.

There is available little information on the annual cost of emergency repairs for concrete stoppings. One operator reports that 10 per cent of such stoppings are each year affected by heavy shots and require repairs amounting to 50 cents each. Most operators find no occasion for emergency repairs. A nearby windy or blown-out shot, or a general mine explosion, is usually the only extraordinary agent that destroys such stoppings; and even then the wreckage from them is small and is easily cleaned up, and most of the blocks are suitable for reuse. When the bottom is soft and has a tendency to heave, it has been found advisable to lay the foundation course of cement blocks at a right angle to the other blocks in order that a larger bearing surface may be obtained.

The methods and appliances used in the manufacture of cement blocks at the mine are described by S. O. Andros in Bulletins 4 and 6 of the Illinois Coal-Mining Investigations.

Plate I, *B*, shows an efficient cement-block stopping.

LINE BRATTICES.

In a few mines in southern Illinois, where some of the entries are being driven into gaseous territory, line brattices are constructed in order to conduct the ventilating current from the last crosscut to the working face. In mine-rescue and mine-recovery work it is often necessary to use line brattices to expedite the movement of a body of afterdamp. Except for such work, line brattices are seldom used in Illinois coal mines.

Plate III, *B*, shows the outer or crosscut end of a line of brattice in the Bureau of Mines experimental mine, at Pittsburgh, Pa., and Plate IV shows the inner or face end of the same brattice. When a line of brattice cloth is run from the last crosscut to the face of an entry that contains explosive or noxious gases, a miner should nail the canvas on the intake side to avoid danger from the gases.

ROCK-DUST-PROTECTED STOPPING.

A rock-dust-protected stopping has been designed by George S. Rice, chief mining engineer of the Bureau of Mines, for checking explosions that may pass through a crosscut. In one explosion test at the bureau's experimental mine it prevented an explosion from entering the butt headings, and in other tests, when placed in crosscuts, it prevented the explosion in one butt from passing into the parallel butt entry. The construction of the stopping and method of erection are fully described in Technical Paper 84[a] of the bureau.

The stopping is a double one, as shown in figure 4. The principle is that when pressure from a slight explosion is thrown on the stop-

[a] Rice, G. S., and Jones, L. M., Methods of preventing and limiting explosions in coal mines: Technical Paper 84, Bureau of Mines, 1915, pp. 32–35.

A. ROCK-DUST-PROTECTED STOPPING.

B. OUTER OR CROSSCUT END OF LINE BRATTICE, BUREAU OF MINES EXPERIMENTAL MINE.

INNER OR FACE END OF A LINE BRATTICE.

ping, half of it goes out and the other half is left in position with some dust on the shelves to extinguish a delayed flame. In case of a very violent explosion, the whole stopping goes out at once and the rock dust extinguishes the flame. This type is adapted for use with any kind of a solid stopping as a core, or it may be used by itself.

The construction is simple and is not expensive at mines where fine inert dust is available; such as fine ashes, or the fine screenings from limestone quarries, which, in Illinois, can be purchased for $1 to $2 per ton. The curtains prevent the dust from becoming damp under ordinary mining conditions. In especially wet mines it may

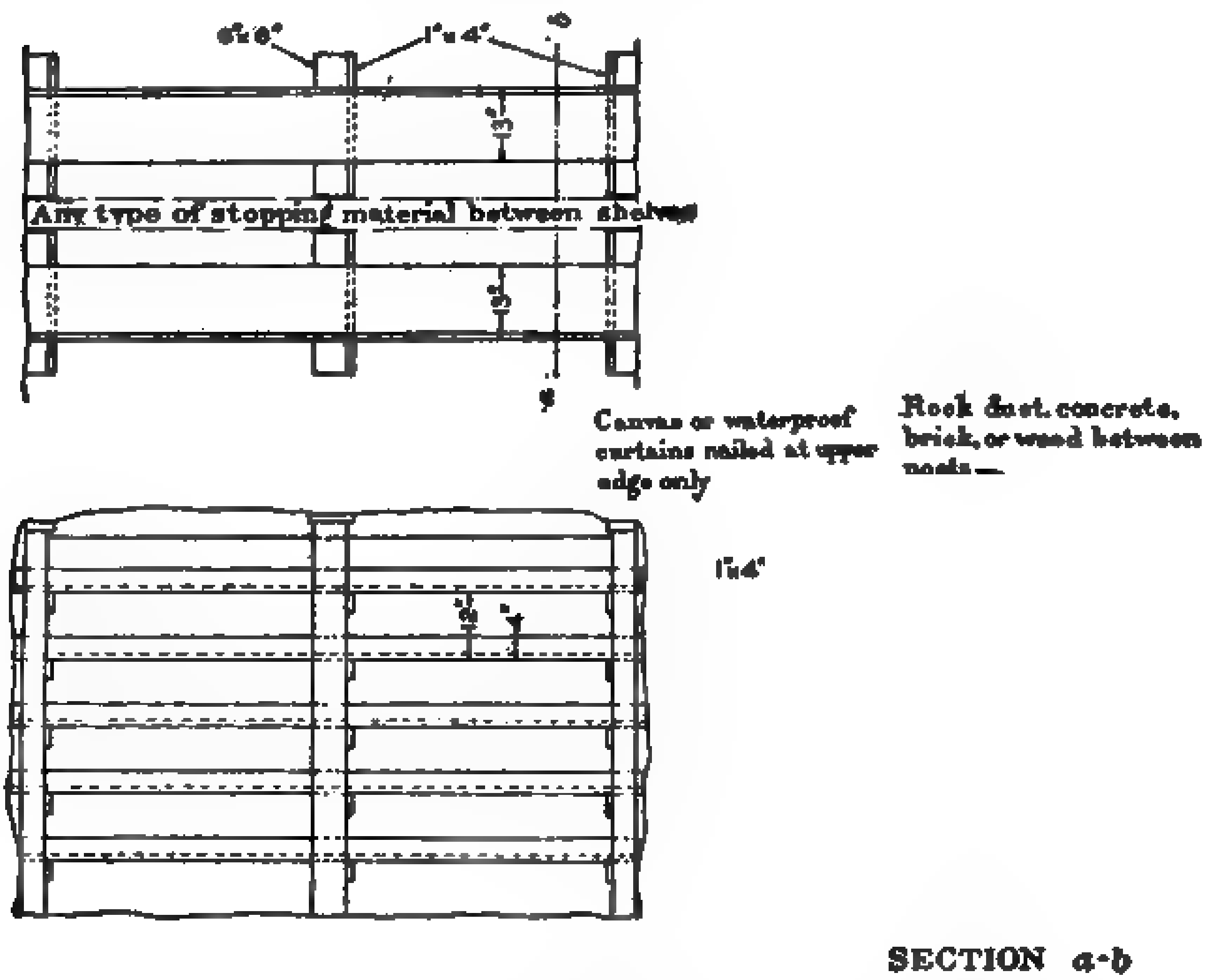

FIGURE 4.—Rock-dust-protected stopping designed by Bureau of Mines.

be necessary to put waterproofing over the stopping so that the water may be deflected to one side or the other. Galvanized-iron sheets may be used or the walls may be plastered with neat cement.

Plate III, *A*, shows the rock-dust-protected stopping as erected at the bureau's experimental mine.

COLLAPSIBLE STOPPING.

A great deal has been written about the use of collapsible stoppings which would go out at the instant of an explosion and thus relieve the pressure of the explosion wave. The writer believes that the arguments for and against the use of these stoppings are not

conclusive, and that the initiation of an explosion should be prevented by the use of permissible explosives, by efficient humidification, and by ample ventilation to prevent the accumulation of pockets of methane, rather than that recourse should be had to a flimsy stopping which will permit the leakage of air required for the well-being of the men at the face. If it is thought best to use a collapsible stopping the rock-dust stopping previously described accomplished the purpose sought and also furnished means of extinguishing that branch of the explosion that blew down the stopping.

CONCLUSIONS.

One of the most obvious conclusions from the inspection of several hundred stoppings is that the efficiency of a mine stopping to prevent the leakage of air depends more on the care with which the joints are made than on the material that is used in construction. Some concrete stoppings were found to leak large quantities of air because they were not set into the rib, and new tongue-and-groove stoppings were tight when carefully constructed. The amortised first cost is a small item when compared with the maintenance and efficiency charges, and the cost of cutting a latch in the rib is only a small part of the initial expense. The preparation of grooves in the ribs should therefore always precede the erection of stoppings.

A general protest is being made against high velocities in the ventilating currents in coal mines. The practical facts on which this protest is based are as follows:

1. The amount of power required to move a given volume of air increases directly as the cube of the velocity. If leaky stoppings make it necessary for the fan to deliver 4 times as much air as would be required if the stoppings were tight, the cost of the power to move the larger volume of air would be 64 times that for the smaller volume.

2. As the temperature and the relative humidity of the upcast at any mine are approximately constant throughout the year, the quantity of moisture extracted from the dust will vary directly with the quantity of air in circulation. If therefore the fan furnishes 4 times as much air as would be required if the stoppings were efficient, the amount of water extracted from the mines during cold weather would be increased nearly fourfold. The drying out of coal mines in the winter causes a dangerous condition to exist in the event of a blown-out or windy shot or the ignition of a pocket of gas.

3. The weight of fine dust that is in suspension in the mine air varies approximately as the cube of the velocity of the ventilating current. This relation has been established by W. H. Carrier, chief engineer of the Buffalo Forge Co., in experiments to determine the

weight of material which can be handled by velocities above the critical speed. The greater the velocity of the air, therefore, the larger will be the amount of fine coal dust that will be in suspension and deposited evenly along the entry way ready to assist in the propagation of an explosion throughout the mine.

It is evident that mine economy and mine safety require that stoppings along the main air passageways should be efficiently installed and maintained.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF COAL MINING.

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of especial interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls., 7 figs.

BULLETIN 50. A laboratory study of the inflammability of coal dust, by J. C. W. Frazer, E. J. Hoffman, and L. A. Scholl, jr. 1913. 60 pp., 95 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 56. First series of coal-dust explosion tests in the experimental mine, by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 60. Hydraulic mine filling; its use in the Pennsylvania anthracite fields; a preliminary report, by Charles Enzian. 1913. 77 pp., 3 pls., 12 figs.

BULLETIN 62. National mine-rescue and first-aid conference, Pittsburgh, Pa., September 23–26, 1912. By H. M. Wilson. 1913. 74 pp.

BULLETIN 68. Electric switches for use in gaseous mines, by H. H. Clark and R. W. Crocker. 1913. 40 pp., 6 pls.

BULLETIN 69. Coal-mine accidents in the United States and foreign countries, compiled by F. W. Horton. 1913. 102 pp., 3 pls., 40 figs.

BULLETIN 72. Occurrence of explosive gases in coal mines, by N. H. Darton. 1915. 248 pp., 7 pls., 33 figs.

BULLETIN 82. International conference of mine-experiment stations, Pittsburgh, Pa., U. S. A., September 14–21, 1912, compiled by G. S. Rice. 1914. 99 pp., 4 figs.

BULLETIN 83. The humidity of mine air, with especial reference to coal mines in Illinois, by R. Y. Williams. 1914. 69 pp., 2 pls., 7 figs.

BULLETIN 87. Houses for mining towns, by J. P. White, 1914. 64 pp., 8 pls., 9 figs.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 12 pp.

Technical Paper 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

Technical Paper 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

Technical Paper 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

Technical Paper 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

Technical Paper 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

Technical Paper 15. An electrolytic method of preventing corrosion of iron and steel, by J. K. Clement and L. V. Walker. 1913. 19 pp., 10 figs.

Technical Paper 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

Technical Paper 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

Technical Paper 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

Technical Paper 21. The prevention of mine explosions, report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

Technical Paper 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

Technical Paper 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

Technical Paper 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

Technical Paper 39. The inflammable gases in mine air, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 2 figs.

Technical Paper 43. The effect of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

Technical Paper 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

Technical Paper 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

Technical Paper 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp., 10 figs.

Technical Paper 52. Permissible explosives tested prior to March 1, 1913, by Clarence Hall. 1913. 11 pp.

Technical Paper 58. The action of acid mine water on the insulation of electric conductors; a preliminary report, by H. H. Clark and L. C. Ilsley. 1913. 26 pp., 1 fig.

Technical Paper 69. Production of explosives in the United States during the calendar year 1912, compiled by A. H. Fay. 1914. 8 pp.

Technical Paper 71. Permissible explosives tested prior to January 1, 1914, by Clarence Hall. 1914. 12 pp.

Technical Paper 75. Permissible electric lamps for miners, by H. H. Clark, 1914. 21 pp., 3 figs.

Technical Paper 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

Technical Paper 77. Report of the Committee on Resuscitation from Mine Gases, by W. B. Cannon, George W. Crile, Joseph Erlanger, Yandell Henderson, and S. T. Meltzer. 1914. 36 pp., 4 figs.

TECHNICAL PAPER 78. Specific-gravity separation applied to the analysis of mining explosives, by C. G. Storm and A. L. Hyde. 1914. 13 pp.

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Bulletin 100

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

VAN. H. MANNING, ACTING DIRECTOR

MANUFACTURE AND USES OF ALLOY STEELS

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HENRY D. HIBBARD

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INTRODUCTION.

The object of this report is to give briefly information of present value relating to the manufacture and uses of the various commercial alloy steels, with the hope of stimulating the demand for such steels and extending their practical use. The report is issued by the Bureau of Mines as a contribution to the increase of efficiency in the preparation and utilization of the mineral resources of the United States.

Alloy steels are included in the so-called special steels, but as the latter term is often used in the mills to designate broadly any steels intended for purposes other than those served by the regular product, it has seemed best to use the more specific term of alloy steels in this report.

Alloy steels are bringing about a series of revolutions in various industrial fields in which steel plays an important part. Most elements that could be procured in sufficient quantity have been alloyed with iron in various proportions, either alone or in combination with others, in the search for useful alloy steels. Those steels that have gained and maintained for themselves a place in current use are discussed in this report. Some of them have had an ephemeral life of usefulness which would no doubt have been prolonged had not some other more satisfactory steel been developed.

Probably the first useful alloy steel was Mushet's self-hardening tungsten tool steel, patented in 1868. Fifteen years later chromium steel, really containing chromium, was struggling for recognition for some purposes, the chief of which was for the manufacture of solid shot for piercing armor. In both of these steels the effect of the alloying element as used was in a way proportional to the amount contained. In 1882 Hadfield made his epoch-making discovery of manganese steel and demonstrated that in iron metallurgy it is not safe to take for granted anything as to the properties of an alloy of iron with other elements, basing one's opinion on past experience and knowledge, and that the effect of an alloying element may not be proportional to its content. The development of useful nickel

steels followed in a few years and the field thus opened has since then been worked by many able and zealous men, with results of great importance and value.

DEFINITIONS.

Definitions of terms used throughout this report are presented below:

Simple steel, often called "carbon steel," consists chiefly of iron, carbon, and manganese. Other elements are always present, but are not essential to the formation of the steel, and the content of carbon or manganese, or both, may be very small.

Alloy steel is steel that contains one or more elements other than carbon in sufficient proportion to modify or improve substantially and positively some of its useful properties.

Simple alloy steel is alloy steel containing one alloying element, as for example, simple nickel steel.

Ternary steel is alloy steel that contains one alloying element, the term being synonymous with "simple alloy steel."

Quaternary steel is an alloy steel that contains two alloying elements such as chromium-vanadium steel.

Complex steel is an alloy steel containing more than two alloying elements, such as high-speed tool steel.

Alloy-treated steel is a simple steel to which one or more alloying elements have been added for curative purposes, but in which the excess of the element or elements is not enough to make it an alloy steel.

Raw steel is steel as cast, either an ingot or casting.

Natural steel is steel in the condition left by a hot-working operation, and cooled in the open air.

Normalized steel is steel that has been given a normalizing heat treatment intended to bring all of a lot of samples under consideration into the same condition.

Annealed steel is steel that has been subjected to an annealing operation.

Hardened steel is steel that has been hardened by quenching from or above the hardening temperature.

Tempered steel is steel that has been hardened and subsequently tempered by a second lower heating.

These definitions are based on the definition of steel that states that steel must be usefully malleable. The definitions of alloy steels do not include effects which are negative, or the prevention or cure of ills which the steel might possess were the alloying element or elements not added.

An iron alloy is not herein considered as useful unless it presents some useful property or modification of a property not offered to the same degree by a simple steel.

The definition of alloy steel given does not agree with that of all writers on the subject of mixtures of iron with other elements than carbon; but it does agree with that of some who have been careful enough when considering the whole range of elements to designate them as alloys, such as silicon-iron alloys and chrome-vanadium alloys, the range covering the useful alloys or steels as well as those in which the alloying element is added for curative purposes and others that have only a scientific interest.

Elements other than carbon may be desired in steel, and therefore be added to or permitted to remain in it for three distinct purposes, as follows:

1. To give the composition desired and to cure in simple steels some ills or defects that the final product might otherwise possess.

2. To make alloy steels. Such elements are manganese, silicon, tungsten, nickel, chromium, vanadium, cobalt, and others of less importance.

3. To make alloys which, though they excite only a scientific interest, form a great part of the whole field of iron metallurgy. Many a one of these alloys would have a commercial value if another alloy were not known that meets particular requirements more satisfactorily either as to efficiency or as to cost or both.

The various additions of the element manganese to iron illustrate well the three purposes of alloys as above specified. A moderate amount, usually less than 1½ per cent, is added to molten steel made by an oxidation process (pneumatic or open hearth) to prevent red-shortness. A much larger amount is added to make commercial manganese steel, which should contain 11 to 14 per cent of manganese. Outside of these limits, a great number of manganese-iron alloys may be made, most of which have only a scientific value, though the number of useful ones is always liable to be extended as new requirements and methods of manufacture and treatment arise. Simple commercial steels containing between 1.5 and 2 per cent of manganese are made, and manganese steels containing less than 11 per cent manganese are useful for certain purposes; but the manganese content of the great bulk of the steels made is within the limits given.

The total number of possible alloys of iron with varying proportions of other elements is of course practically infinite. So, indeed, is the number of useful alloys, though they form only a small fraction of the whole number.

This report deals exclusively with alloy steels, as defined above, in which the alloying element or elements modify directly, positively, and usefully some of the properties of the products.

LIST OF USEFUL ALLOY STEELS.

The eight alloy steels named below in the chronological order of their introduction are considered to meet this requirement:

1. Simple tungsten steels.
2. Simple chromium steels.
3. Manganese steel.
4. Simple nickel steels.
5. Nickel-chromium steels.
6. Silicon steels.
7. High-speed tool steels.
8. Chromium-vanadium steels.

The first four and the sixth of these are ternary steels, the fifth and eighth are quaternary, and the seventh is of complex composition. Some of these steels may be treated while molten by the addition of a purifying or solidifying element or elements in such a small quantity as not greatly to affect the final properties. Thus a small amount of titanium, aluminum, or vanadium may be added to a chromium or nickel steel and hardly appear in the final analysis. Such a result is seen in the alloy-treated steels.

ALLOY-TREATED STEELS.

Alloy-treated steels need be only briefly alluded to here. The method of manufacturing all steels made by the oxidation processes involves the presence in suspension or solution of harmful amounts of oxygen or oxides, and before the metal is cast means must be taken to lessen the oxygen or oxide content to or below an allowable maximum. One or more of certain elements having at steel-melting temperatures a stronger affinity for oxygen than iron has, are added to the molten metal. The oxygen leaves the iron to seize such added elements, forming new products insoluble in the iron, which in time are precipitated, gather together, and leave the metal.

Such unfinished steel also contains in solution a quantity of gases which require to be decomposed or kept in solution, for, if not, when the steel is solidifying, part of the gases will leave it and part will be imprisoned in the metal and form gas holes, of the variety commonly called blowholes. The addition of certain elements to the metal tends to prevent the separation of the gases.

Further there is a tendency of certain of the ingredients of steel to collect or segregate in an injurious way in the upper central part of a large ingot or casting, but the addition of certain elements lessens this tendency.

Elements that are added to prevent, minimize, or cure these ills are manganese, which is the most important, silicon, aluminum, titanium, vanadium, and others of less importance. The effects

aimed at are therapeutic and though real and valuable, are mostly negative rather than positive; that is, effort is made to cause the steel to be free from some or all of the defects cited.

The proportions of these elements added are, generally speaking, only enough to cure the defects to be removed or counteracted, with a suitable excess to reasonably assure such a result in view of the uncertainties and irregularities of steel making. The excess of any of the elements named has indeed some effect on the final properties of the steel, but not enough to put the product in the class of the alloy steels. Steels so treated are considered as alloy-treated steels, they being simple steels, outside the subject of this paper, as the alloying elements do not give new or modified properties of important commercial value.

Some alloying elements are added to simple steel in such proportions as to produce only a curative effect, and the product can not definitely be classified as a simple or an alloy steel, as can be done when a rather large excess of the element is added. The elements vanadium and silicon are examples, both being added to cure ills in the steel, and an excess of either causes the physical properties of the steel to vary to some extent. Both are used also in undoubted alloy steels that have unique properties, which would not be anticipated from the observation of the effect of a moderate excess of either in a simple steel.

Crude alloys of the alloying elements that are used as ingredients in steel making and are not useful themselves in their crude state are not herein considered.

THE USES OF ALLOY STEELS.

With few exceptions all alloy steels are heat treated for use, the treatment developing in them the high physical properties they are capable of possessing. No general law regarding the effects of heat treatment of alloy steels can be laid down. Some steels when quenched from a high heat are hardened and others are softened, the latter being generally those with the higher contents of certain of the alloying elements. In respect to the effects of heat treatment each steel is considered by itself.

Developments in the manufacture of alloy steel and in the heat treatment of steel have occurred somewhat simultaneously during the past 30 years, and care is needed lest the benefits gained from one be confounded with those afforded by the other. The highest merit is obtained from the adoption of both developments together—that is, the use of heat-treated alloy steels. Usually heat treatment has contributed more to the superior properties of the metal than has the use of alloys.

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The alloy steels discussed in this report are considered as regards their value for structural, cutting, or electrical purposes.

Steel used for structural purposes is taken to include that used for the stationary as well as the moving parts of structures and machines, including bridges, buildings, vehicles, machine tools (except the cutting tools), armor plate, ships, ores involving resistance to abrasion or corrosion, and wire, except electrical wire, and in general all steel not used in the other two fields.

Steel used for cutting purposes includes that employed to form an actual cutting edge and that used in projectiles for war.

Steel for electrical purposes is used in magnets, core steel, nonmagnetic articles, and electrical-resistance devices.

No steel suitable in a commercial sense for two of these purposes is made, though some steels might be used for more than one purpose if a better kind for the other specific purpose were not known; thus a fair tool steel might be made of some of the harder structural steels, and a fair magnet might be made of some of the tool steels.

The effects of the alloying elements in alloy steels are various; thus nickel increases the elastic limit as compared to tensility; chromium increases the hardness of quenched steel; and manganese destroys magnetic susceptibility—effects all of which are valuable for certain purposes.

MANUFACTURE OF ALLOY STEEL.

Alloy steels are made by any of the steel-making processes, that is, by any of the variations of the pneumatic processes, by the acid or basic open hearth, by the electric furnace, and by the crucible. For each of the various purposes, however, the practice is more limited, the general rule being of course that the cheapest process is employed that will yield a product satisfactory for the purpose in view.

All alloy-steel ingots or castings should be made sound and with full tendency to pipe. Soundness, which means freedom from gas holes, is, generally speaking, a necessary requirement in order that the product may be sound, as almost any of the alloying elements interferes with or prevents the welding of steel that contains that element so that any contained gas holes in ingots will not be welded up by hot working. Therefore they may, if near the surface, be opened to the air by scaling in heating or by forging and rolling and then be oxidized within and form seams. Chromium and nickel more than other alloying metals prevent welding in steel.

Pipe in the ingot may be shortened in length by casting the ingot with the larger end up, or it may be avoided if the ingot when so cast is squeezed laterally, or if the top is maintained in the molten state until the remainder of the ingot is solid. If neither of these means

is employed enough of the top of the ingot must be discarded to get rid of any objectionable pipe. Whether any pipe is permissible depends on the use to be made of the steel, there being many uses for which steel containing a pipe is adapted, or a hole may even be drilled where the pipe would naturally be as is often done to favor heat treatment in massive articles.

The amounts given in this report of the different steels produced in the United States have for the most part been determined by indirect means. They are therefore not exact, but are, it is believed, near enough to the truth to warrant presenting them.

The temperatures are given in both centigrade and Fahrenheit degrees.

Alloy steels and other alloys of iron with other elements have been discussed by numerous writers chiefly in their general or purely scientific aspects.

The selected bibliography, at the end of each chapter, relating to the different alloy steels is intended to be limited to articles that bear upon the useful steels. Many of the articles themselves have bibliographies more or less complete. The bibliography is arranged chronologically.

STRUCTURAL ALLOY STEELS.

GENERAL CONSIDERATIONS.

Structural steels, whose fields of use have already been noted, have some attributes in common, which makes it worth while to consider them collectively to a certain extent. These steels are working great improvements in the production of structures for various purposes, especially where the saving of weight or increase of strength or both are important, the most conspicuous example being undoubtedly the automobile industry. Heat-treated alloy steels with double or treble the strength of the simple steels they replace and with as great or greater reliability are now in regular and most advantageous use. In common with other alloy steels, structural alloy steels owe a part of their superior properties to the presence of the alloying element but usually far more to heat treatment when it can be given to them. In automobiles the use of alloy steels is generally not advised unless the steels are heat treated, as the gain from their use in the natural or untreated state does not compensate for the increased cost.

Most structural alloy steels are therefore used in the heat-treated condition, when the articles made of them are, like automobile parts, not too bulky or massive. Large pieces like nickel-steel rails and nickel-steel members of bridges are used without heat treatment, the advantages of increased strength and ductility that the metal possesses being due solely to the presence of the alloying element.

The difficulties attending the heat treatment of large steel parts that are bulky for their weight are holding back their general introduction. They require, as nearly as is practicable, to be uniformly heated, uniformly cooled in quenching, and afterward, when cold, to be made true to form, as the quenching operation, however carefully done, usually leaves them warped or twisted. No doubt, in time, means will be found to overcome these drawbacks and such pieces as rails, and bridge members. of alloy steels will be used regularly in the heat-treated condition. A compact object like an armor plate, though very large, may be quenched without unmanageable warping because of its simple shape. The difficulty in making straight and true such an article as a heat-treated rail of pearlitic alloy steels lies largely in the springiness of the treated metal. It is not easy to give it the correct amount of set needed to counteract or obliterate a crook, bend, or twist that may result from quenching. Yet this is necessary when the piece must be straight or true to shape. Stretching slightly beyond the elastic limit as is done to some thin steel sheets and relatively small bars to straighten them might be efficacious, but is not to be easily done with a piece of such irregular cross section as a rail.

The effects of heat treatment are so great that a certain steel may be given a very wide range of properties, depending on the treatment, and any desired set of properties within that range may be obtained solely by varying the heat treatment. The principal variant is the degree of the second heating. The lower this is, the stronger and stiffer the steel, and the higher, the weaker and more ductile it is.

This effect of heat treatment on steel is illustrated by a table published by a producer giving the results of 40 tensile tests made from one heat of steel, each test piece having had a different heat treatment. Five, which cover the range, are given in the table below.

Results of tensile-strength tests of 5 pieces of steel each receiving a different heat treatment.

Tensility.	Elastic limit.	Elastic ratio.	Elongation in 2 inches.	Contraction of area.
Pounds.	*Pounds.*	*Per cent.*	*Per cent.*	*Per cent.*
84,850	50,500	60	28	67.5
120,975	90,000	74.5	14.5	51
166,950	157,500	94	12.5	44
205,600	200,000	97	13	48.7
240,975	225,000	93	9	20.5

Analysis of the original steel showed C, 0.25; Mn, 0.50; Cr, 1.07; and V, 0.17 per cent, but similar results could be obtained with a variety of compositions.

For making small parts that must be true and well finished the structural alloy steels are generally heat treated before they are machined, and this requirement prevents the use in such parts of

steel of the highest strength attainable because steel having that strength is not commercially machinable. Generally speaking, any part that is to have an elastic limit of more than 100,000 pounds per square inch must be treated after having been machined, not before, because most steels having a higher elastic limit than that are too hard to allow machining by commercial processes, though chromium-vanadium steels with an elastic limit of 150,000 pounds per square inch are claimed to be machinable, that is, they may be cut with high-speed steels at a profitable rate. An elastic limit of 100,000 pounds or more per square inch can be imparted to steel only by heat treatment, as no untreated steel of a commercial grade will have so high a limit.

Some of the makers of structural alloy steels are publishing for each of their steels a graph showing the physical properties the steel will have when hardened and then drawn to different temperatures. Of course, the graphs give ex parte information which is subject to confirmation before acceptance, but the plan is excellent as giving the most information in the least space. Similar graphs of many alloy steels prepared by consumers are expected to be soon available for comparison. From these graphs a new user of these steels may choose the properties he desires and specify the steel he wishes, making some allowance of course (say 10 per cent) for the uncertainties of manufacture and treatment. The steel maker or treater, to be reasonably sure of meeting the requirements, will aim to exceed the properties specified, and the net result will usually be that the steel will have practically the properties desired.

The tables of properties given later show how much more the properties possessed are imparted by treatment than by composition.

The size or massiveness of the article has a great effect on the results obtained by any given heat treatment. The greater the mass the lower the qualities, though not in exact proportion. Thus the mass must always be considered in connection with the properties desired, and the composition and heat treatment prescribed must be modified accordingly, though even then the effect of mass may be only partly compensated for.

The modulus of elasticity of many, if not all, structural alloy steels in common with other steels is not changed much by heat treatment or variations in composition [a] and is usually between 28,000,000 and 30,000,000 pounds per square inch; that is, the modulus of the steel in its annealed, hardened, and tempered condition remains practically unchanged. The following table was compiled from data given by Landau.[b]

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[a] Landau, David, Influences affecting the fundamental deflection of leaf springs: Bull. Soc. Automobile Eng., vol. 5, March, 1914, p. 430.

[b] Landau, David, op. cit., pp. 431-434.

Moduli of elasticity of some alloy steels.

Composition of steel.								Modulus.
C.	Si.	Mn.	P.	S.	Cr.	Ni.	V.	
Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	
0.50	0.13	0.82	0.01	0.02	1.25	....	0.14	29,240,000
.47	1.83	.70	.01	.01	....	....	....	28,950,000
.48	.16	.44	.01	.01	.98	2.02	....	28,840,000
.30	.19	.64	.01	.01	....	3.25	.18	28,260,000
.25	.21	.74	.01	.01	....	3.55	....	28,170,000
.24	.21	.46	.01	.02	.96	2.02	....	28,200,000
.25	.16	.50	.01	....	1.05	....	.16	30,158,000

Because of the unchangeability of the modulus of elasticity the stiffness or rigidity of steel within the elastic limit is not changed either by heat treatment or the presence of any of the alloying elements, except perhaps manganese in manganese steel and nickel in high-nickel steels.

Heat treatment does increase the elasticity, however, so that a piece of heat-treated steel may return to its original form after having endured a stress that would have permanently deformed it in its untreated condition; that is, it is given some of the springiness of heat-treated springs.

Many of the structural steels, particularly those used in automobile manufacture, have a great endurance against fatigue when subjected to repeated alternating stresses. The heat treatment increases their durability in this test even more noticeably than it does the properties determined by the tensile test.

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SIMPLE TUNGSTEN STEEL.

Mushet's air-hardening steel, the first of the alloy steels, may be considered as a simple tungsten steel though it contained so much manganese (about 2 per cent) that it might with some reason be classed as a quaternary steel, as it contained also about 2 per cent of carbon and 6 per cent of tungsten. The manganese was essential to give the self-hardening property. The steel is now becoming obsolete, having had a useful career and having formed an indispensable link in the development of the high-speed tool steels discussed later.

Tungsten is very heavy, its specific gravity being, according to recent determinations, about 19.3, and it is the most infusible substance known except carbon and, perhaps, boron. These properties have some effect on the production of tungsten steel.

MANUFACTURE OF TUNGSTEN STEEL.

Tungsten steel is generally, if not always, made by the crucible process. The pots are charged cold by packing in the materials, the tungsten being placed at the top to counteract in a measure its tendency to settle because of its high specific gravity. If this tendency operated unchecked there might be at the bottom of the pot a rather infusible mush of high-tungsten alloy which would not pour out, and if it did the ingot would have an irregular composition because of the uneven distribution of the tungsten.

The steel is melted and then "killed" in the crucibles by holding them in the furnace for 30 or 40 minutes after the charge has melted, until the steel ceases to bubble or work and lies dead in the pot.

The pots are sometimes cast singly or doubly by hand pouring or collectively by means of a ladle into which all the pots of a furnace charge are emptied. Good tungsten steel makes remarkably sound

solid ingots, except for the pipe, though tungsten itself is not considered to aid in removing or controlling either the oxides or the gases. It is added solely for its effect on the finished and treated steel.

This lack of power of tungsten to deal with oxides and gases arises no doubt from its low calorific power, its heat of combustion being given (with qualification) as about 1,000 calories, whereas iron burned to Fe_3O_4 gives 1,612 calories.

METHOD OF WORKING.

Simple tungsten steels of commercial grades are heated, forged, and rolled in much the same manner as other high-carbon steels, presenting no special problems or difficulties.

PROPERTIES AND USES.

Simple tungsten steel is at present chiefly used in permanent magnets for electric meters, in small dynamos, and hand use, for which it has been used for 30 or 40 years. The consumption in 1913 is thought to have been between 5,000 and 6,000 tons. This steel contains about 0.6 per cent of carbon and 6 per cent of tungsten. Some has been made in recent years containing 0.2 to 0.3 per cent of vanadium, chromium, or molybdenum which were considered at the time to give greater retentivity to the steel, but those ingredients are now generally held to be of no practical value, adding nothing to the fitness of the steel for its purpose.

Some buyers of magnet steel do not specify composition but only performance, that is, what magnetic properties the steel must have.

To make permanent magnets retain their magnetism as much as possible they are made very hard by heating and quenching. They are then magnetized; and if they are to be used for electric meters they are seasoned by a treatment involving protracted heating to 100° C. (212° F.) to make their magnetism as nearly constant as possible.

A variety of tungsten steel containing about 1 per cent of carbon and 3 to 4 per cent of tungsten is made and used as a tool steel for taking finishing cuts on iron and steel in the machine shop. It acts more like a simple steel than a self-hardening steel, as it requires to be hardened by quenching in water and then drawn in the same general way that simple steels have been drawn, presumably for thousands of years. It will cut at a higher speed than a simple steel, say 40 feet per minute on steel having a tensile strength of 80,000 pounds per square inch, and is also more durable.

The presence of tungsten in steel is generally stated to lower the fusion point of the steel. Mars[a] gives a table of fusion points of

[a] Mars, G., Die Specialstähle; ihre Geschichte, Eigenschaften, Behandlung und Herstellung. Stuttgart, 1912.

tungsten steels with contents of tungsten ranging from 0.5 to 17 per cent, from which he concludes that tungsten lowers the fusion point. However, when his results are corrected for the lowering effects of the contained carbon, silicon, and manganese doubt arises as to the correctness of his conclusion. Thus, a steel containing 0.66 per cent C, 0.03 per cent Si, 0.04 per cent Mn, and 3.11 per cent W fused at 1,488° C. The carbon would lower the fusion point about 60° C., and the silicon and manganese slightly, so that the plain iron-tungsten alloy should have a fusion point a little above 1,548° C., which is about 20° C. above that of pure iron. Seemingly this is the effect of 3.11 per cent tungsten.

The erosion of the bore of cannon by the powder gases is held to depend largely on the fusion point of the metal of the tube or liner, the higher the point the greater being the resistance to erosion. So it has been found that the nearer the metal comes to being pure iron the higher its fusion temperature and the better it resists erosion, but the strength required compels a certain amount of hardening and strengthening elements to be present in the steel. Tungsten raises the strength and possibly the temperature of fusion and so has been employed for the tubes of cannon, particularly by the Government of Austria. Arnold and Read [a] found that steel with 0.71 per cent carbon and 5.4 per cent tungsten had in the annealed state a tensility of 88,900 pounds per square inch, an elastic limit of 60,200 pounds, an elongation of 20 per cent, and a contraction of area of 34.7 per cent, values that compare favorably with those of the steels usually employed in the manufacture of cannon.

They give data regarding a series of annealed tungsten steels as follows:

Data regarding annealed tungsten steels.

Composition.							Tensile properties.				Condition when turned.
C.	W.	Si.	Mn.	P.	S.	Al.	Tensility.	Yield point.	Elongation in 2 inches.	Contraction of area.	
P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Pounds.*	*Pounds.*	*P. ct.*	*P. ct.*	
0.73	2.4	0.11					84,200	48,100	20.5	31.5	Moderately tough.
.71	5.4	.11					88,900	60,200	20.0	34.7	Tough.[f]
.70	9.7	.04					126,100	90,000	14.0	22.1	Very tough.
.73	15.0	.03	(b)	(c)	(d)	(e)	98,500	....	25.0	43.3	Do.
.72	21.1	.06					104,300	57,300	20.5	39.2	Very tough, slightly hard.
.67	26.3	.06					110,600	....	9.0	11.4	Do.

[a] Arnold, J. O., and Read, A. A., The chemical and mechanical relations of iron, tungsten, and carbon, and of iron, nickel, and carbon: Proc. Inst. Mech. Eng., pt. 2, March-May, 1914, pp. 223-248.

[b] 0.15 or less.

[c] 0.02 or less.

[d] 0.04 or less.

[e] 0.01 or less.

[f] Tough means that the lathe chips curled off in spirals.

The strength and hardness of these steels may be greatly increased by heat treatment involving quenching and with only relatively small decrease in ductility.

THEORY OF TUNGSTEN STEEL.

Arnold and Read concluded that the carbon in the steels they examined was combined with iron when the tungsten was low, but that the higher the tungsten the more of the carbon was combined with it until in steel containing 11.5 per cent of tungsten none of the carbon was combined with iron, but all of it with tungsten. With still higher tungsten content the excess of tungsten was combined with iron.

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SIMPLE CHROMIUM STEEL.

Simple chromium steels, though one of the earliest if not the first of the alloy steels to be made, are not now largely used though they are for quite a variety of purposes. In combination with other alloying elements, however, chromium is still one of the most important constituents of alloy steels.

The production of simple chromium steel ingots is thought to have been about 6,000 tons in 1913 with a content of chromium of 0.4 to 2 per cent. It is made by either the acid open hearth or crucible process. If made in crucibles the chromium in the form of ferrochromium is made a part of the original charge, but if made in the open-hearth furnace the ferrochromium is added just long enough

before casting for the alloy to be melted and well mixed with the charge.

The consumption of ferrochromium for steel making in this country in 1913 was between 3,000 and 4,000 tons, containing 60 to 70 per cent of chromium. If 1 per cent be assumed as the average content of chromium in the structural steels, and 4 per cent in the high-speed tool steels, a total of about 204,000 tons of steel of all kinds in the ladle contained chromium. The bulk of such steel was quaternary or complex steel, about 6,000 tons being considered as simple chromium steel, as already stated.

At steel-melting temperatures chromium has a greater affinity for oxygen than has iron, and therefore if any oxygen either free or in the form of oxide of iron comes in contact with melted steel that contains chromium, some of the chromium is wasted. It is not oxidized as rapidly as silicon and manganese, however, and therefore has little deoxidizing effect when added to an oxidized iron bath, the oxides and gases being controlled by other means.

The effect of a chromium content up to a maximum of 2½ per cent in steel is to increase the hardness moderately when the steel is in the natural state, as defined on page 6, and particularly when it is in the hardened condition after having been quenched.

METHODS OF WORKING.

Chromium steels are cast, forged, and rolled by the same plant and by the same methods as simple steels of the same or slightly higher carbon contents. Castings are annealed, or heat treated, as the conditions warrant or require to give the most suitable properties for the proposed use.

Chromium steels are perhaps never used in the untreated condition, and their properties in that state are therefore not given.

Composition and properties of heat-treated simple chromium steels.

Sample No.	Constituents.						Tensile strength.	Elastic limit.	Contraction of area.	Elongation in 2 inches.	Ball hardness.	Heat treatment.	
	C	Mn	Si	S	P	Cr						Temperature at which steel was quenched in water.	Temperature at which hardness was drawn in air.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Pounds.*	*Pounds.*	*P. ct.*	*P. ct.*		*°C.*	*°C.*
1	0.70	0.54	0.09	0.01	0.01	0.70	129,000	121,700	60	21	235	816	593
2	.70	.54	.09	.01	.01	.70	110,900	105,300	63	26	195	816	649
3	.70	.54	.09	.01	.01	.70	88,000	73,000	68	36	168	816	754
4	.40	.78	.54	.02	.01	.92	143,500	131,600	56	18	242	816	538
5	.40	.78	.54	.02	.01	.92	103,200	90,200	69	26	201	816	714
6	.91	.35	.08	.03	.01	.91	96,800	69,300	63	28	175	....	....

USES OF SIMPLE CHROMIUM STEELS.

The longest established use of chromium steels now current is in stamp shoes and dies for pulverizing certain gold and silver ores. These shoes and dies contain 0.8 to 0.9 per cent of carbon, with 0.4 to 0.5 per cent of chromium. They are preferably annealed to destroy ingotism and so impart some toughness to the metal, which increases their durability in an important degree.

Another long-established use of chromium steel is in 5-ply plates for the manufacture of safes for the safe-keeping of valuables. These plates are made of five alternate layers, two of chromium steel and three of soft steel or wrought iron, and after having been hardened offer great resistance to the drilling tools employed by burglars. The plates are, however, necessarily rather thin, usually between one-half and 1 inch thick, so that a safe wall or door to be more than an inch thick must be formed of two or more thicknesses, which are fastened together with screws. Many portable safes so made have been robbed by burglars who used liquid nitroglycerine, which was flowed into the joints and exploded, blowing out the door or wall. Such safes are therefore not made in such great numbers as formerly, but considerable quantities of chromium steel are used in large stationary safes, usually called vaults, where the individual pieces are larger and the other safeguards against burglary so effective that they are not attacked by burglars.

Hardened chromium-steel rolls having 0.9 per cent of carbon and 2 per cent of chromium are used for cold-rolling metals. They are glass hard so that the ball hardness can not be determined, the ball making no impression. The hardness, as determined by the sclerescope, is 107.

Files of chromium steel are excellent, and in 1913 about 3,000 tons of this steel went into them, the carbon content being 1.3 to 1.5 per cent and the chromium content about 0.5 per cent.

An important use of chromium steel is in balls and rollers for bearings. One large maker uses steel containing carbon, 1.10 per cent; chronium, 1.40 per cent; manganese, 0.35 per cent; sulphur, 0.025 per cent; and phosphorus, 0.025 per cent. Sizes smaller than one-half inch diameter are heat-treated by being quenched in water from 774° C. (1,425° F.) and then drawn to 190° C. (375° F.) for half an hour. For larger balls the quenching temperature is 802° C. (1,475° F.). The second heating does not produce an oxide color, but is enough to let down in some degree the internal stresses due to the irregular cooling of quenching so that the balls are less liable to crack spontaneously or to be broken in use.

The strength of a good, well-treated ball is prodigious, a ball three-fourths of an inch diameter, tested by the three-ball method,

sustaining a load of 52,000 pounds. On the small area of contact the intensity of the pressure amounts to over one million pounds per square inch.

The Society of Automobile Engineers recommends less chromium than that given above, or 1 to 1.2 per cent. A steel maker recommends 1.5 per cent.

A former important use of these steels was in armor-piercing projectiles. Cubillo [a] gives as the analysis of such a steel: C. 0.85 per cent, Mn. 0.38 per cent, Cr. 2.31 per cent, Si. 0.16 per cent, and P. 0.02 per cent. The physical properties were as follows:

Physical properties of certain armor-piercing steels.

Treatment.	Elastic limit.	Tensility.	Elongation in 2 inches.
	Pounds.	*Pounds.*	*Per cent.*
Annealed	47,700	102,000	20.5
Heat-treated	79,400	114,500	18.5

The greater part of projectiles, however, contain some nickel also, as noted in the discussion of nickel-chromium steels.

Chromium steels are preferred by some builders in certain automobile parts, though most makers of cars prefer nickel chromium.

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MANGANESE STEEL.

Manganese steel in the commercial meaning of the name is a variety of iron containing 11 to 14 per cent of manganese and 1.0 to 1.3 per cent of carbon. The original patents covered alloys of iron with 7

[a] Cubillo, L., The manufacture of armor-piercing projectiles: Jour. Iron and Steel Inst., vol. 88, 1913, p. 251.

to 30 per cent of manganese but the steels within the limits of composition given have the greatest strength and ductility of any and are always meant when manganese steel is ordered without further qualification. Departure from these limits of analysis means in a general way that the steel is to that extent unsuited for the structural purposes for which manganese steel has been found applicable.

The bulk of the manganese steel made at present is put into castings, of which about 36,000 tons was made in 1913. The use of hot-worked (rolled or forged) manganese steel is now of importance, some 3,000 tons having been made in 1913, nearly all of which went into rails. These quantities call for about 60,000 tons of molten steel in the ladle.

Though about 40 concerns offer manganese-steel castings, only about half a dozen make any considerable quantity, having supplied practically all of that made in 1913.

MANUFACTURE.

Manganese steel is still made in the ladle according to Hadfield's expired patents by the mixture of decarburized iron and 80 per cent ferromanganese. The decarburized iron is prepared either by the pneumatic process, being blown in some one of the many modified pneumatic converters, or in the Siemens furnace. As ferromanganese forms such a large proportion of the charge, about one-seventh, it must be melted or nearly so before being added to or mixed with the decarburized iron, or the resulting steel would be too cold. The ferromanganese is usually melted in graphite crucibles or pots which is an expensive way but not yet generally superseded. The pots are heated in ordinary coal-fired melting holes. They usually may be used to melt eight or ten charges, whereas in steel melting they average less. This longer life when melting ferromanganese comes from both the lower melting temperature needed and the absence of solution of the graphite of the pot by the metal, common ferromanganese being saturated with carbon, whereas molten steel eagerly dissolves any carbon with which it comes in contact. The ferromanganese need not be completely melted, which of course would not be permissible with the steel.

The decarburized iron used should have a low carbon content, not over 0.10 per cent, so that the resulting steel may not have more than is desired. The proportion of carbon to manganese in the steel will, therefore, be a little more than in the ferromanganese.

After the manganese steel has been made in the ladle it should be cast as soon as practicable if it is to be used for castings, but if it is to be used for ingots a little time should be allowed for the silicate formed within the metal to collect and float to the top.

The quantity of manganese is proportioned to the size of the charge of decarburized iron with allowance for loss through oxidation of an amount equal to about 1½ per cent of the steel. Thus 14 per cent is added to yield 12.5 per cent in the steel. Everything is weighed—the stock before melting, the iron added to the vessel if decarburized by the pneumatic process, the molten ferromanganese, which is put in the ladle first, and the decarburized iron are all weighed, the weight of the decarburized iron giving the weight of the molten steel in the ladle. When the decarburized iron is prepared in a tilting Siemens furnace, a part may be taken out at a time and made into steel, the amount of iron being weighed by a suspended weighing machine on the crane.

Earnest efforts have been made to melt ferromanganese in a cupola because of the high cost of melting in pots, and some has been so melted in a commercial way. When the ferromanganese is melted in a cupola there is more loss of manganese, which offsets, in part at least, the lower cost of melting. The loss is diminished to a large extent by charging the ferromanganese in the center of the cupola with coke around the outside so that the metal is largely protected from free oxygen of the blast which would oxidize manganese if it came in contact with it. It is a matter for calculation which is to be chosen for any given conditions as being the cheaper or, all things considered, the most advantageous.

Another effect of the waste of manganese from the ferromanganese used in making manganese steel is that the carbon content is higher and perhaps undesirably so in the finished steel because of its concentration due to the waste of manganese. Carbon is not oxidized under usual conditions in fused ferromanganese because of the presence of the large excess of manganese which seizes any oxygen that reaches the alloy.

Manganese steel is usually made and handled in clay-lined ladles. Basic-lined ladles present some advantages, but clay is used because it is so much cheaper and more easily made into a lining. Such a lining is of course to be classed as acid, and sometimes strongly so, when quartz in the form of sand or ganister is mixed with the clay. Molten manganese steel in an acid-lined ladle continually makes a fusible and liquid slag, which is chiefly a silicate of manganese, green in color. It is formed of oxide of manganese, which is continually being oxidized by atmospheric oxygen, and the ladle lining. A little green silicate rises and joins the slag from within the metal, being formed from the silica contained in the decarburized iron and the oxide of manganese formed by substitution of manganese for iron in the oxide of iron in the metal. The thin fluid slag is troublesome, as it is not easily kept from entering the molds with the steel, where it is likely to cause a defect in the ingot or casting.

To avoid the trouble from the very liquid slag a "coffeepot" ladle is sometimes used, which is provided with a spout like that of a coffeepot, attached at one side and connecting with the ladle interior at the bottom. This arrangement acts effectively to skim the slag, which can not enter the spout until nearly all the steel has been poured out of the ladle.

The corrosion of the ladle lining is deepest at the slag line when the ladle is full, as the slag has more time to act on the lining at that level than at any other.

Because of its large content of carbon, silicon, and manganese, the latter fusing at 1,260° C., manganese steel melts at about 1,325° C., a temperature lower than that of simple steel, and one that favors the running of intricate castings. For the same reason manganese steel, containing so much gas solvent, is usually free from gas holes; but if the decarburized iron of which it is made is too hot, and therefore too heavily charged with gases, the solvent powers of the silicon and manganese may be exceeded, and the steel be saturated with gases, the ingots or castings being consequently infested with blowholes by the gases liberated in cooling.

COMPOSITION.

In making manganese steel one composition is practically standard. The usual analyses of manganese steel lie between the following limits: Carbon, 1.0 to 1.3 per cent; silicon, 0.3 to 0.8 per cent; manganese, 11.0 to 14.0 per cent; phosphorus, 0.05 to 0.08 per cent. The sulphur content is so low as to be negligible in manganese steel as in other iron-manganese alloys, from which any sulphur that may get in is quickly eliminated by the manganese, probably as sulphide, which rises to the surface or enters the slag. In the open the sulphur is burned by the oxygen of the air, forming sulphurous acid (SO_2), which may often be smelled coming from the molten steel.

Low-manganese steels with 7 to 8 per cent of manganese are finding some use, having a higher and better defined elastic limit than the regular grade and yet with considerable though much less ductility. They are also cheaper to make. They do not flow under impact as freely as the regular commercial grades, because, no doubt, of their martensitic structure.

Manganese-iron alloys containing 3 to 10 per cent of manganese and 1 per cent of carbon are martensitic. With the manganese over 10 per cent the structure is austenitic. The steels having 7 to 10 per cent of manganese are so different from commercial manganese steel that another name should be given them to avoid confusion. The name "loman steel," an abbreviation of "low-manganese steel," has been applied to them and seems to be suitable as a short distinctive name.

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GENERAL PROPERTIES.

Manganese steel is a hard self-hardening steel, owing this property to its composition and not to treatment. It can not be softened by heating followed by slow cooling. It is, for a metal, a poor conductor of electricity.

Manganese steel has a high coefficient of expansion, small patterns being made with a shrinkage of five-sixteenths of an inch to 1 foot, which sometimes is not quite enough. A shrinkage of five-sixteenths of an inch to 1 foot gives a mean coefficient of expansion of about 0.000024 per degree centigrade.

In respect to specific gravity manganese steel is not to be distinguished from simple steels of the same carbon content, as all have, generally speaking, about the same.

Perhaps the most remarkable property of manganese steel is its almost total lack of magnetic permeability and susceptibility. This metal, containing 85 per cent of iron in a metallic form, is so slightly attracted by a magnet that the pull can not be felt with the hand, whereas magnetic oxide of iron, containing about 70 per cent of iron in a nonmetallic form, is strongly attracted. A magnetometer or even a pocket compass needle will usually detect magnetism in commercial manganese steel, especially if a test be made soon after the steel has been in contact with a magnet, but the amount that may be so detected is extremely small. For most practical purposes it is zero.

PROPERTIES OF MANGANESE STEEL IN THE RAW STATE.

The properties of manganese steel in the raw state are much like those of other raw high-carbon steels, the metal being very hard, but its ductility being practically negligible. The steel, because nonmagnetic, may be used for purposes requiring a hard nonmagnetic metal, if it is not liable to shock. With better tool steels, which may make machining manganese steel a commercial operation, the metal may find a field of use in electrical apparatus, replacing some brass or other nonferrous metals because of its lack of magnetic qualities and lower cost.

HEAT TREATMENT OF MANGANESE STEEL.

Although the composition of manganese steel is extremely important in determining its properties, the heat treatment to which it is subjected to develop in it its great toughness or ductility is even more so.

As used, it is almost universally water-toughened according to the method Hadfield set forth in his early papers[a] on the subject. This

[a] Hadfield, R. A., On manganese steel: Jour. Iron and Steel Inst., 1888, pt. 2, pp. 41-82; Manganese in its application to metallurgy: Proc. Inst. Civ. Eng., vol. 93, pt. 3, 1887-88, pp. 1-16; Some newly discovered properties of iron and manganese, loc. cit., pp. 61-126.

treatment consists in heating the whole article to about 1050° C. and then cooling it as quickly as possible by immersing it in cold water, the colder the water and the more of it, the better. It will not do to heat only a part of the piece for quenching, and if a part of a toughened article becomes heated to redness or near it by accident or design the whole piece should be reheated and again quenched to give it proper qualities for use.

No time should be lost in completing the heating and quenching after the piece has become red-hot to avoid oxidation as completely as possible. Manganese steel is a poor conductor of heat, a factor that interferes with its heat treatment and tends to limit the thickness of the steel that may be profitably treated. This limit of thickness is generally taken as 4 inches, though somewhat thicker pieces in which the presence of internal cracks in the central parts would not be ruinous are treated in particular instances.

A thick piece of manganese steel must be heated at a slow rate. The heating can not be properly done by placing the piece when cold in a hot furnace. The furnace if hot from previous use must be cooled down nearly to atmospheric temperature or to that of the piece to be toughened, if it is not cold, before it is placed in the heating chamber. If the furnace is much hotter than the piece the poor heat conductivity of the metal retards the passage of heat to the interior, and the temperature of the skin rises rapidly, because of which and because of the high coefficient of expansion of the metal the exterior is expanded so much sooner than the interior that the latter is likely to be torn apart, the cracks extending outward from the center toward the surface until the stresses are sufficiently relieved or they reach metal too much softened by the heat to crack.

Many have asked when first considering the question, "Why not quench the casting when still hot with the initial casting heat?" The answer is that the heat of the various parts of the piece is not sufficiently uniform, the thicker parts being hotter than the thinner. If the piece is what might be called of uniform thickness, any corners or edges or parts first laid bare after casting are cooler than the rest of the piece, and even if the piece be a sphere, which is almost the only shape that will show a surface of uniform temperature after casting, the interior will be so much hotter than the exterior, and will, therefore, when the sphere is immersed in a cooling bath, continue contracting so long after the surface is cold and contraction there has ceased that it will tear itself apart. The toughness of the metal will not prevent internal rupture under such conditions as there can be no contraction of area, as in the pulling test, to compensate for and so permit stretching of the metal, which must therefore separate at its weaker spots and be subject to cracks.

PROPERTIES OF HEAT-TREATED MANGANESE STEEL.

Hadfield's papers of 1888[a] gave the results of a large number of physical tests of manganese steels. These results indicate that the toughening effect of water quenching was imparted only when the steel contained at least 9 per cent of manganese. Since that date steels having as little as 7 per cent of manganese have been given a useful toughening effect though only a fraction of that of commercial manganese steel.

The hardness of toughened manganese steel is unique, and it may be termed a tough hardness and not a flinty hardness. Such steel may easily be dented with a hammer or marked with a file or chisel, but cutting it to a useful extent is almost impracticable, so that such finishing as is necessary is usually done by grinding with abrasive wheels.

The water toughening of manganese steel gives it great ductility—greater as to elongation, perhaps, than that of any other steel and exceeding sometimes 50 per cent in 8 inches, although its high degree of hardness is not greatly altered. This high ductility in combination with the great hardness of manganese steel gives it great resistance to abrasive wear as well as safety from breakage. Practically all manganese steel is used in the toughened state.

In the pulling test the percentage of contraction of area is less than the elongation, a result directly opposite to that with simple as well as most alloy steels, in which the percentage of contraction is usually twice or more that of the elongation. The pulled test piece has a rather uniform stretch throughout its length, whereas simple steels, as is well known, have a largely increased amount of stretch near the point of fracture. When a piece of manganese steel is pulled, the increase of strength due to cold working (stretching) is greater than the decrease in cross section due to contraction, so that a stretched part becomes stronger than the unstretched parts, and elongation then occurs at another place. As the pulling is continued, all parts of the pulled section stretch one after the other, with the result that when the piece is finally ruptured the stretch has been comparatively uniform. There is indeed an increased local extension and contraction close to the point of rupture, as with other ductile steels, but it is less marked in the manganese steels.

The elastic limit of manganese steel is unexpectedly low and not well defined, as the steel yields at a gradually increasing rate when pulled, as in testing, giving no point that, strictly speaking, can be said to be the elastic limit or even yield point. The strain diagram has no jog.

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[a] Hadfield, R. A., loc. cit.

A recent pulling test of forged, heat-treated, manganese steel gave the results following. The steel was cast in a test bar 3 inches square, forged down to a test piece of about the dimensions given, and finished by grinding.

Results of pulling test of piece of manganese steel.

Diameter of piece, inches	0.823
Length, inches	2
Tensile strength per square inch, pounds	152,840
Elastic limit per square inch, pounds	56,400
Elongation, per cent	51
Contraction, per cent	39.5
Carbon, per cent	1.10
Manganese, per cent	12.4
Silicon, per cent	0.15
Phosphorus, per cent	0.06

The length of the pulled section of a manganese-steel test piece does not affect the elongation as much as is the case with simple steels because the stretch is so much more nearly uniform, as described above.

Owing to its lack of elastic limit and to its high ductility, manganese steel is prone to flow under stress, and it does not have high resistance to compression or to continually repeated blows of a hard mineral or other material that will gradually batter it out of shape. The results of tests of this property made with the Government testing machine at the Watertown Arsenal were as follows:

Results of compression tests of cast manganese steel.

No. of test piece.	Analysis.				Permanent set at a pressure per square inch of—				Total load.
	C.	Si.	Mn.	Cr.	40,000 pounds.	50,000 pounds.	60,000 pounds.	70,000 pounds.	
	Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Pounds.*
1	1.23	0.95	12.6	....	0.0006	0.0036	0.0213	0.0981	190,100
2	1.26	.54	12.8	....	.0020	.0046	.0182	.0899	180,100
3	1.31	.43	12.7	....	.0010	.0036	.0204	.0998	172,300
4	1.22	.72	11.7	0.86	.0002	.0009	.0038	.0220	175,200

All test pieces were cast and finished by grinding to 4 inches long and 1.129 inches in diameter, giving 1 square inch of cross-sectional area. At the total load the pieces buckled. The permanent set at a pressure of 40,000 pounds per square inch shows that the limit of elasticity was passed in every case.

The hardness by Brinell's ball test of manganese steel is low, running usually about 190.

The merit number obtained by multiplying the figure representing the tensile strength of a material by the figure representing its

elongation gives roughly a fair idea of the amount of work that must be expended upon the material to break it—that is, its strength must be overcome through a distance represented by its ductility before it will be broken. The merit number of manganese steel is perhaps the greatest of all known steels. The merit numbers of various metals are given below.

Merit numbers of various metals.

Metal.	Tensile strength.	Elongation.	Merit number.
	Pounds.	*Per cent.*	
Manganese steel	140,000	50	7,000,000
Soft steel	60,000	30	1,800,000
Tool steel	130,000	5	650,000
Cast iron	20,000	.5	10,000
Nickel steel, natural	95,000	21	1,995,000
Nickel steel, heat-treated	207,000	14	2,898,000

Of course, each of these metals yields to stress within its elastic limit, so that some work must be expended on it to strain it up to that point. With a nonductile metal, such as cast iron or even raw manganese steel, this is the reason for such small resistance to shock as it has. The elongation of 0.5 per cent given in the table is high for cast iron, but its power to absorb energy without breaking is increased by its ability to be strained within the elastic limit. If a material were to have no ductility and were perfectly rigid, the slightest blow would break it. Glass, having no ductility and being nearly rigid, comes near to being such a material. Of course, it may be easily broken by a blow, though it offers considerable resistance to a static load—enough to be useful, as in a floor plate, for example.

The effect of the toughening operation on the merit number of manganese steel is shown by the following results, taken from Hadfield's first paper.[a]

Results of tests of toughening operation on merit number of manganese steel.

Sample No.	Condition.	Tensile strength.	Elongation.	Merit number.
		Pounds.	*Per cent.*	
1	As forged	88,120	3.5	308,420
	Toughened	145,240	50.0	[a]7,262,000
2	As forged	81,600	1.56	127,296
	Toughened	150,370	44.44	[b]6,682,440

[a] The toughened steel was 24 times as hard to break as the untoughened.
[b] The toughened steel was 50 times as hard to break as the untoughened.

MANGANESE-STEEL CASTINGS.

Manganese-steel castings are made in dry sand, in green sand, and in some instances in iron molds, the considerations leading to the adoption of any particular material being much the same as with

[a] Hadfield, R. A., Manganese in iron and steel: Proc. Inst. Civ. Eng., vol. 93, pt. 3, 1887-88, pp. 1-16.

ordinary simple steel castings, such as danger of pulling apart or cracking in cooling, misrunning, or failure to fill the mold properly, and breaking or washing of the mold, and numerous others. The high coefficient of expansion of manganese steel must be considered, as it increases the liability of a casting to be cracked or pulled apart by shrinkage in cooling.

Manganese steel is prone to settle as it solidifies, demanding, for a given massiveness of design, larger sink heads than simple steels to feed the casting properly and prevent settle holes.

Even more than with other steel castings it is important that manganese-steel castings be so designed that the mass is fairly uniform throughout, or in particular that no part is much thicker than the rest. If a thick part is unavoidable, it should be connected with a sink head by metal as thick or nearly so. Thus, bosses and heavy fillets, often advisable in iron and simple steel castings, should be avoided because of the local increase of the mass they cause. The trouble is that a heavy part incompletely fed will be unsound in its central parts. A hole or recess cored in, if permissible, may prevent the central cavity, or an iron or soft steel core may be imbedded in the thick part, which, by hastening the solidification of the metal, may prevent the formation of holes or loose metal there.

USES OF MANGANESE STEEL.

The specific uses for which manganese steel is employed may be learned from the catalogues of the producers. It is used in equipment employed in the mining, milling, and treatment of ores and other mineral products, quarrying and rock-dressing, digging and dredging, railway tracks, parts of machines exposed to gritty wear, and burglar-proof safes and vaults for the safe-keeping of money and valuables.

Because of its low-yield point manganese steel does not give satisfaction in many lines for which otherwise it seems to be eminently fitted.

When the service required is such that the resistance of the metal to flow is not exceeded, the results are often excellent. When for any reason the pressure at the point of wear passes the flowing point the results are disappointing, though even then they may be better than those obtained by the use of any other material.

In applying manganese steel to a new use determination of its fitness is nearly always a matter to be demonstrated, though this is less general than formerly as knowledge and experience have increased. Some of the fields that it was thought at first would be well occupied by manganese steel it has never won; others that were formerly unthought of it has found easy of conquest. Much uncertainty has come in applying it to new uses by the requirement that

manganese steel as a metal be substituted for some other metal of a certain size and shape. The results are often not fully foreseen and liable to be unsatisfactory as such a substitution is itself unscientific. The proper way to apply a new metal of unique properties to a new use is to study the service to be rendered and then to devise methods of doing the work and to design parts therefor such as will utilize to the best advantage the superior properties of the new metal. Sometimes however this is not easily done. Nevertheless manganese steel, because of its properties, presents difficulties that are usually best overcome by such a plan if the highest benefit attainable from its use is to be gained.

It is a well-established rule that manganese steel resists admirably abrasion under slow speeds of impact as in Blake crushers, rolls, gyratory crushers, and similar machines but results in high-speed grinders, such as the various centrifugal mills, are, if not poor, at least such as will not often warrant the expense of manganese-steel wearing parts, especially if such parts require some finishing, which must be done by slow and expensive grinding. In accordance with the same rules manganese steel makes good car wheels that are to run at slow speeds such as on mine cars, but it is unsatisfactory in wheels used on railway cars that are run at high speeds.

For railway-track work manganese-steel cast frogs, switches, curved rails, and other special work are most excellent and they are extensively used.

The properties of the metal were early seen to make it an ideal material of which to make burglar-proof safes and vaults; that is, it is too hard to be cut, and too strong to be broken even by considerable charges of dynamite and nitroglycerin. It has now been employed for this purpose some 15 years and no genuine manganese-steel safe has yet been opened by burglars though a few have been attacked. Burglars have learned the futility of their efforts to rob such a safe.

The nonmagnetic property of manganese steel has found an important use in the cover plates of lifting magnets for handling heavy iron and steel articles where it is subjected to hard blows from the pieces jumping to meet the magnets. It offers little or no obstruction to the passage of the magnetic attraction. It is also used in the structure about the compasses on some ships because it does not affect the compass needle. This use may become important.

On the magnetic survey yacht *Carnegie* of the Carnegie Institution, which is built wholly of wood, bronze, and other nonmagnetic materials, manganese steel, because of its nonmagnetic property, is used in the grate and other parts of the gas producer which were necessarily made of steel.

HOT-WORKED MANGANESE STEEL.

Manganese steel is, like simple steel, or even more so, improved in its physical properties by hot working (forging or rolling). Cast test pieces usually give misleading results because of imperfections due to casting. To cut out of the solid and finish by grinding a test piece of cast manganese steel is expensive, and when done the piece is liable to be so imperfect as to give practically valueless information. In this respect, however, it is like many other cast metals.

A steel that cast and heat-treated may show a tensile strength of 80,000 pounds per square inch with 20 per cent elongation may have, when well worked by forging and rolling and then heat-treated, a tensile strength of 140,000 pounds per square inch and 50 per cent of elongation in 8 inches.

The first commercial use of manganese steel was in dredge pins for chain bucket dredges, which were forged from square ingots in 1889 or 1890 by Hadfield at Sheffield. The greater ease of casting the metal, however, led to the employment of castings wherever they could be used. The records made by castings have stimulated in recent years the production of rolled pieces, particularly rails, plates for chute linings, and screens for crushed stone. A beginning has been made in the use of forged manganese steel for purposes for which its great strength and ductility (its merit number) and the consequent margin of safety it affords make it particularly reliable. Its use in spring hangers on locomotives is an example. Cold working such as stretching or cold rolling rapidly raises its tensile strength and elastic limit but destroys most of its ductility. Cold-rolled manganese steel on test has shown a tensile strength of 250,000 pounds per square inch and an elastic limit of 230,000 pounds.

For screening coke woven screens of manganese-steel bars are giving a promise of life a hundred times as great as that of screens made of the soft simple steel usually used for this purpose. Coke rapidly wears away metal on which it impinges, as though it were composed of or contained many minute diamonds.

The high electrical resistance of manganese steel would no doubt lead to its general use as a resistance material in the form of wire, for which it has been tried, were it not that its hardness makes drawing it into wire too costly an operation. Its electrical properties are good for this purpose, its specific resistance running from 65 to 75 microhms per cubic centimeter.

Agricultural implements made largely of rolled plates, such as shovels and hoes, offer a most inviting field for the use of manganese steel, especially for kinds of service in which the plates are used

up by wear. If the service is such that they rust or corrode much more than they wear, tools made of simple steel may be as good. This field is now being entered.

The largest demand for hot-worked manganese steel is in rails for railroads. The rails are rolled on ordinary rail mills and are heat-treated by being quenched immediately after rolling. The service rendered by the rails is excellent and their use is extending. Some railroad men think their durability at least five times that of ordinary rails. Their value to the railroads is not, however, as great as their service compared with that of simple steel rails as the interest on the extra price and the relatively small value of the scrap when they are worn out tell against them, though the saving in the cost of laying is in their favor.

Perforated plates of manganese steel for screening ores, crushed stone, and other mineral products are promised, but special punching machinery is needed to punch the holes, as the holes probably must be punched at a much slower speed than that at which simple steels are punched.

CONCLUSIONS.

The economies resulting from the use of manganese steel in its various fields of service have been great. Perhaps the most conspicuous example of its use was in connection with the Panama Canal where years of time and millions of expense were saved by its use.

The promise of usefulness because of its great tensile strength and more remarkable ductility, there is ground to hope, may in the near future be fulfilled.

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SIMPLE NICKEL STEELS.

Nickel steel was chronologically the fourth alloy steel to be introduced, it having been in use for 25 years, and the steels to which nickel is added aggregate a large tonnage. In 1913, including the nickel-chromium steels as well as the simple nickel steels, the total amount of steel produced to which nickel was added was about 150,000 tons in the ladle, of which about 50,000 tons was simple nickel steel. The field for the latter is being steadily narrowed by the substitution of the cheaper or better nickel-chromium steels.

The useful nickel-iron alloys range, with large intervals, from 2 to 46 per cent of nickel, a greater compass than is covered by any other element alloyed with iron. The addition of less than 2 per cent of nickel alone does not seem to give enough benefit to make the addition worth while.

Nickel in untreated ordinary nickel steel raises the tensile strength, and in a greater proportion the elastic limit for a given content of carbon without decreasing the ductility.

Nickel steels with the different percentages of nickel present about the same range of internal microscopic structures as do manganese-iron alloys. With low-nickel content, as in the great bulk of nickel steels made, the unhardened steel is pearlitic. Higher nickel content gives martensitic structure and still higher austenitic. Certain steels with each of these structures find uses as is noted in subsequent pages.

MANUFACTURE OF SIMPLE NICKEL STEEL.

Nickel steel is made by any of the steel-making processes but most of it is produced in the open-hearth furnace. The operations are similar to those followed in the production of simple steels,

the nickel being either in the materials of the original charge or added in the metallic form at any time long enough before the heat is cast for the nickel to be melted and thoroughly mixed with the metal of the charge. Nickel is negative to iron at steel-melting temperatures and the iron protects it from oxidation and even reduces it from its oxide so that it is not wasted to any considerable extent in melting or working even when iron ore is added to the bath. On the other hand it does not deoxidize the metal or decompose carbonic oxide or keep the hydrogen and other gases in solution. It is not added, therefore, for curative purposes as it gives no aid in rendering steel sound, or free from holes. In fact, nickel steel is prone to have seams and surface defects after it has been rolled, which is one reason against its wider use. The service of nickel is merely as an alloying element, to improve the physical properties of the finished steel either in its natural or heat-treated condition.

As might be expected from an alloying metal whose atomic weight, specific gravity, and fusion point are so near those of iron, nickel does not segregate much as the steel solidifies, and is claimed to hinder in some degree the segregation of carbon and the other less metallic ingredients. A reasonable explanation of this action of nickel on the other elements has not yet been established.

WORKING OF SIMPLE NICKEL STEEL.

Ordinary simple nickel steel (3 to 4 per cent nickel) is worked hot by the usual forging and rolling operations much as simple steel is worked. The higher nickel steels are more difficult to work, having narrower ranges of temperature at which they may be hot-worked without showing signs of redshortness. In the ordinary grades seams and adhering scale give some trouble.

Although molten iron protects molten nickel from oxidation, as stated, iron can not protect nickel from oxidation in scale formed on nickel steel, as in the heating furnace. The scale formed sticks much more firmly to the metal than that of simple steel both hot and cold and requires particular measures for its removal. Articles such as plates, having large flat surfaces from which the scale can not escape even if loosened by such means as rolls or the flat dies of the forging press, are sometimes cleaned cold by electrically driven machines which break up the scale by a shower of blows with chisels or hammers. A round article is much more satisfactorily cleaned of scale by the forging operation, as the metal is worked while not in contact with the dies, and the scale is thereby loosened and falls off. The scale naturally escapes more easily from the underside of a rolled plate than from the top in rolling, and some mills are

equipped with apparatus for turning the plate bottom up before it is finished, so that the scale which has been repeatedly loosened and then rolled in again by the rolls can drop off when loosened from the under side.

Steels containing useful quantities of nickel are liable to contain seams that appear as dark-colored lines in the metal. The seams doubtless come, sometimes at least, from "skin" gas holes which become oxidized on their walls, and, although such oxidized holes will, if present, forms seams in any steel, they seem to persist more in nickel steel because, perhaps, the nickel prevents the welding of such holes, as may happen with holes in simple steels if they are squeezed together while hot enough and the walls are clean and not oxidized. A hole near the surface in nickel steel might conceivably, therefore, be drawn out and the slit formed be opened to the air by the hot working, then oxidized on its inner surface, and form a seam, when a similar hole in simple steel would be welded and therefore not form a seam. Nickel-steel ingots should therefore be made sound and free from gas holes. It is held by some persons that seams develop in rolling without being caused by gas holes, and that this tendency is lessened by rolling at a high temperature, about 1,300° C. (2,372° F.).

GENERAL CHARACTER OF NICKEL STEEL USED FOR STRUCTURAL PURPOSES.

The great bulk of simple nickel steels contain from 2 to 4 per cent of nickel, a proportion that affords the most suitable physical properties for nearly all structural purposes, and the nickel content usually aimed at in steels for structural purposes is 3.25 per cent. This grade might be called ordinary nickel steel as it is usually meant when nickel steel is mentioned without further specification. It has high value for structural purposes such as bridges, gun forgings, machine parts, engine and automobile parts, and any similar line of service that is too severe for simple steels.

The bridges in which it is used are particularly those of great span, and it is nearly always used in the natural or annealed condition when the additional strength and ductility imported is that due to the mere presence of nickel in the metal. Important quantities are used in the Queensboro, Manhattan, St. Louis Municipal, and Quebec bridges, and the Kansas City viaduct. Some nickel-steel tension bridge members have been heat treated by heating and quenching, being immersed in water edgewise with the longitudinal axis horizontal, and afterward drawn back by a second heating to give an elastic limit of 55,000 pounds per square inch, a rather low figure. A relatively low percentage of nickel, or about 2 per cent, is sufficient to afford steel with such a property, when heat treated.

The use of nickel steel in bridges saves some weight, a detail of importance in such bridges as those mentioned, but when the span is moderate a bridge of simple steel is perhaps as good and is less costly even though it contains considerably more weight of metal.

Steel with 2 per cent of nickel is used in seamless tubes such as are used for bicycles and for other equipment requiring a high-grade tube. They are not heat treated, but higher properties than those of the steel in its natural state are imparted by the cold-drawing operations by which these tubes are finished. The ordinary grade with 3.5 per cent nickel is used in cannon, being always heat treated for this use. It is also used in many automobile parts, the variety of high properties obtainable in it by modifying its heat treatment rendering it fit for almost any service demanding a strength and security from breakage that a simple steel will not meet.

In some large dynamos the revolving fields are connected by nickel-steel rings having 3 per cent nickel, the metal being particularly well suited for the purpose both by its strength and its magnetic efficiency, the permeability being high and the hysteresis losses low.

CHARACTERISTICS OF DIFFERENT NICKEL STEELS USED IN RAILS.

Nickel-steel rails usually having about 3.5 per cent of nickel have been tried by many railroads and are generally considered unsatisfactory though small lots are still being made chiefly for use in tunnels and other unusually wet or damp places, both for their ability to resist rusting and for safety from breakage. Their price is nearly twice that of simple steel rails and they sometimes give three times the service but their average life has been much less than that. One lot of Bessemer nickel-steel rails which gave between one to two times the service of simple Bessemer steel rails wore unevenly and the metal flowed over the side of the head on the curve so that it was finally detached in thin splinters, some of which were 3 or 4 feet long. The rails from the upper parts of the ingots were more unsatisfactory than those from the bottom and though the ingots were not examined for their soundness it seems evident that their upper parts were infested with blowholes as well as pipe, and that none of the holes or pipe were welded up in rolling, as the effect of nickel in hindering welding of steel is well established.

PROPERTIES OF ORDINARY NICKEL STEEL.

The properties of ordinary nickel steel are given below. All the samples consisted of small test pieces, and elongations were measured on 2 inches except as noted.

Properties of ordinary nickel steels.

Sample No.	Composition. C.	Mn.	Si.	S.	P.	Ni.	Condition.	Physical properties. Tensile strength.	Elastic limit.	Elongation.	Contraction.	Ball hardness.
	P.ct.	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*		*Pounds.*	*Pounds.*	*P.ct.*	*P.ct.*	
1[a]	0.28	0.57	......	0.03	0.02	3.44	Natural state......	95,420	56,670	[b] 21.2	50	......
2[c]	.40	.64	......	.02	.01	3.43	Annealed..........	98,800	51,400	[d] 12.4	33	......
3[c]	.40	.55	......	.03	.01	3.70	.....do...........	93,180	56,060	[d] 15.8	40	......
4[e]	.20	.65	......	.04	.04	3.5	.....do...........	....	43,000	27	62	170
5[e]	.20	.65	......	.04	.04	3.5	(f)................	....	95,000	20	72	216
6[e]	.20	.65	......	.04	.04	3.5	(g)................	....	140,000	14	61	330
7[e]	.30	.65	......	.04	.04	3.5	Annealed..........	....	63,000	27	63	163
8[e]	.30	.65	......	.04	.04	3.5	(h)................	....	87,000	25	68	207
9[e]	.30	.65	......	.04	.04	3.5	(i)................	....	123,000	15	57	269
10[e]	.30	.65	......	.04	.04	3.5	(j)................	....	187,000	13	57	405
11[k]	.25	.74	0.21	.01	.01	3.55	(l)................	207,000	177,000	14	60	395
12[k]	.25	.74	.21	.01	.01	3.55	(m)................	135,000	117,000	20	67	267

[a] Sample represented untreated steel for Quebec bridge.
[b] In 8 inches.
[c] Full size eyebars for St. Louis Municipal Bridge.
[d] In 18 feet.
[e] Figures taken from Fourth report of Iron and Steel Division: Bull. Soc. Automobile Eng., vol. 4, 1913, p. 168.
[f] Quenched in water at 850° C.; hardness drawn in air at 538° C.
[g] Quenched in water at 800° C.; hardness drawn in air at 316° C.
[h] Quenched in water at 800° C.; hardness drawn in air at 593° C.
[i] Quenched in water at 800° C.; hardness drawn in air at 399° C.
[j] Quenched in water at 800° C.; hardness drawn in air at 316° C.
[k] Figures furnished by Halcomb Steel Co.
[l] Quenched in water at 843° C.; hardness drawn in air at 316° C.
[m] Quenched in water at 843° C.; hardness drawn in air at 538° C.

One per cent of nickel in ordinary nickel steel in the natural state raises the tensility about 6,000 to 8,000 pounds per square inch.

The table shows that ordinary nickel steels may be so made as to have a wide range of properties that make them suitable for any structural purposes for which they are not too expensive.

The properties of one grade of nickel-steel castings made for special purposes are as follows: Composition, C 0.20 per cent, Mn 0.50 per cent, Si 0.35 per cent, Ni 2.50 per cent; tensile strength, 85,000 pounds per square inch; elongation, 25 per cent; contraction, 40 per cent. This steel was not given treatment involving quenching but was merely annealed.

Steel containing 5 to 8 per cent of nickel presents a sort of critical point, that proportion being the lowest at which, with the usual range of carbon, the structure is all martensitic and consequently very hard, the martensitic state being equivalent to the hardened state of simple steels. Such steel is difficult to work hot or cold but can be rolled if proper care is used. It finds some usefulness in places where great resistance to shock is required, particularly in thin shield plates about 0.15 inch thick which are used on one side of the caisson of field artillery to protect the ammunition, and the men who serve it, from rifle fire. A sample analyzed for "Tests of Metals" [a] gave the follow-

[a] Report of the tests of metals and other materials for industrial purposes, 1907, War Department, 1908, p. 42.

ing composition in percentages: C 0.42, Mn 0.49, Si 0.26, S 0.02, P 0.02, Ni 6.68.

The content of carbon determines the minimum amount of nickel which must be present to make the steel wholly martensitic. Thus if the carbon content is low, about 0.20 per cent, 8 per cent of nickel is required, whereas if the carbon content is about 0.80 per cent the steel is martensitic when there is 5 per cent of nickel contained. The analysis given last above represents martensitic steel.

Guillet [a] gives the properties of a similar steel, with 6 per cent of nickel and 0.38 per cent carbon, as follows:

Properties of nickel steel containing 6 per cent of nickel.

Condition.	C.	Ni.	Tensile strength.	Elastic limit.	Elongation.	Contraction.	Shock.
	Per ct.	*Per ct.*	*Pounds.*	*Pounds.*	*Per cent.*	*Per cent.*	
Annealed	0.38	6.0	113,760	99,540	20	65	30
Air hardened at 850° C.			177,750	156,420	11	53	19
Quenched in water			199,080	177,750	10	50	17

He does not say whether this steel was martensitic, but the high elastic limits indicate that it was probably largely so, even in the annealed condition.

Steel with 8 per cent nickel has one transformation point at 510° C. (950° F.) where points Ar_1, Ar_2, and Ar_3 are all merged into one. Eight per cent is the highest useful content of nickel in nickel steel that is amenable to ordinary annealing and quenching operations. Hardening by quenching does not occur in steels containing 10 per cent or more of nickel which are on the contrary softened by heating and quenching.

NICKEL-IRON ALLOY DISCOVERED BY ARNOLD AND READ.

The 13 per cent nickel-iron alloy with 0.55 per cent carbon discovered recently by Arnold and Read [b] is noteworthy as it seems to possess the highest strength of any of the nickel steels. It is so hard as to be unmachinable and the investigators mentioned were not able to drill it even to get some drillings for analysis, the composition mentioned being what they aimed at when making the steel. It has a yield point of about 134,000 pounds per square inch, a tensile strength of about 195,000 pounds, with 12 per cent of elongation in 2 inches. This gives a merit figure of about 2,300,000 which is very high for such a hard steel though it does not compare with the 7,000,000

[a] Guillet, Léon, Nouveaux essais au choc a témperatures variables: Rev. mét., t. 7, October, 1910, pp. 837-844.

[b] Arnold, J. O., and Read, A. A., The chemical and mechanical relations of iron, tungsten, and carbon, and of nickel, iron, and carbon: Proc. Inst. Mech. Eng., March-May, 1914, pp. 223-279.

of forged manganese steel. Steel of this composition might have been expected to show maximum strength as a result of Hadfield's experiments,[a] though he did not include this grade in his series of samples. He found that low-carbon steels with 11.4 and 15.5 per cent of nickel each had a tensility of 210,560 pounds, which was more than was possessed by the steels next above and below. The curve therefore should have reached a maximum between them with a nickel content of about 13.5 per cent.

Arnold and Read's steel should, of course, have a higher tensility, or about 215,000 pounds, to harmonize with Hadfield's, and further tests are needed to establish the exact path of the curve. Arnold and Read note that the composition of this steel nearly corresponds with the formula Fe_7Ni. With such properties as it possesses this steel is likely to find at least a limited field of usefulness.

PROPERTIES OF DIFFERENT STEELS HAVING DIFFERENT PERCENTAGES OF NICKEL.

Before Arnold and Read's discovery of the 13 per cent grade, 15 per cent nickel steel was thought to have the greatest strength of all the nickel steels—that is, in the natural state. This variety has been employed in a few instances for shafting and similar service for which other steels failed, but the amount of it used is negligible in statistics. It is hard to machine, and heating followed by slow cooling does not soften it, though heating and quenching does enough to allow it to be machined slowly. It has a tensility of about 170,000 pounds and an elastic limit of 150,000 pounds per square inch, according to one observer, though, as stated above, Hadfield obtained 210,560 pounds tensility, with little ductility. It is likely that the properties desired when this steel was used, particularly its ductility, could now be surpassed by the much cheaper heat-treated ordinary nickel or nickel-chromium steels.

Eighteen per cent nickel-iron alloy, although not useful, is worthy of note here because of its anomalous action (according to Sexton and Primrose[b]) when cooled from 200° C. (392° F.). At first it contracts uniformly until its temperature falls to 130° C. (266° F.). Then it expands while cooling to 60° C. (140° F.), when it again contracts as the temperature falls farther.

Twenty-two per cent nickel steel is used when resistance to rusting or corrosion is desired. A noted example is the valve stems of the salt-water fire-protection service of the city of New York where the apparatus may not be allowed to become inoperative or hard of action from the formation of rust. It is also used sometimes for the spark

[a] Hadfield, R. A., Alloys of iron and nickel: Proc. Inst. Civ. Eng., vol. 138, 1899, pp. 1-124.
[b] Sexton, A. H., and Primrose, J. S., The metallurgy of iron.

poles in the spark plugs of internal-combustion engines, including automobiles, though commercial nickel wire is more commonly used.

High-nickel steels having 25 per cent or more of nickel and low carbon content (about 3 per cent) are austenitic in structure and in the natural state are softer and tougher than the medium-nickel martensitic steels.

High-nickel steel containing 24 to 32 per cent nickel in the form of wire is used for electrical resistance in small quantity, probably between 5 and 10 tons per year in this country.

The analysis and resistance of samples of Krupp nickel-steel resistance wire are shown below. This wire is used in electric toasters, cookers, irons, and similar devices.

Analysis and resistance of samples of Krupp nickel-steel resistance wire.

Sample No.	C.	Mn.	Si.	S.	P.	Cr.	Ni.	Specific resistance per cubic centimeter.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Microhms.*
1	0.52	0.75	0.10	0.035	0.024	....	30.6	87.9
2	.39	1.00	.70	.035	.025	....	24.2	....

Steel with 27 per cent of nickel is used in bits, stirrups, and spurs in riding harness because of its resistance to rusting. It will nevertheless rust slowly at ordinary temperature under conditions that strongly induce oxidation.

Steels containing more than 24 per cent of nickel are practically nonmagnetic in their ordinary condition, a rather remarkable fact when the high magnetic susceptibility of both iron and nickel alone is considered. The explanation that the critical point marking the change from the nonmagnetic to the magnetic state of iron is lowered by the nickel from about 700° C. (1,292° F.) to below ordinary atmospheric temperatures is, no doubt, sound as far as it goes. When 25 per cent nickel steel is cooled to −40° C. (−40° F.) it becomes magnetic, and retains its magnetism at ordinary atmospheric temperatures. On being heated to 580° C. (1,076° F.), however, the alloy reverts to the nonmagnetic state. This separation of 620° C. between the critical points marking the magnetic states in heating and cooling is great in comparison with the 25° to 50° C. of simple steels, and because of it this steel is classed as irreversible.

The nonmagnetic quality of high-nickel steels is not utilized chiefly because of its capacity for becoming magnetic, as described above, for if it happened to be cooled enough to make it magnetic it could not in most cases be easily demagnetized.

The fact that high-nickel austenitic steels have a somewhat lower modulus of elasticity than the low-nickel or simple steels does not

affect their value for the uses made of them. These steels also have low elastic limits though they are tough and show up well in the shock test. Nevertheless they are generally used not because of superior physical properties but because of their resistance to rusting and corrosion or their electrical resistance. With a carbon content of 0.25 to 0.30 per cent and 32 per cent nickel they are used in valves for gasoline motors with good results.

Nickel steel with 30 per cent of nickel is used in boiler tubes, particularly in marine boilers, for which it is admirable. These tubes are in the natural, not heat-treated state. They resist corrosion better than simple steel tubes and last three times as long. Hence their use is sometimes economical in spite of the much higher cost.

INVAR.

The 36 per cent nickel steel known as Invar is used to the extent of perhaps a few hundred pounds a year in clock pendulums, rods for measuring instruments, and such parts for which its exceedingly slight expansion and contraction when heated and cooled within the atmospheric range gives it a particular value. Nevertheless, its coefficient of expansion, even though small, is not negligible, and compensating means must be used in Invar clock pendulums and in the Invar balance-wheels of watches. A watch with an Invar balance-wheel varied 20 seconds per day during a temperature change of 40° to 90° F., the usual test change, a variation too great for a good watch. Some Invar has as low a coefficient of expansion as 0.0000008 per degree centigrade, and samples have been made that contracted slightly when warmed. The coefficient given indicates an expansion of 0.05 inch in a mile per degree C.

When Invar is heated to about 300° C. (572° F.) and higher its coefficient of expansion is greatly increased and its lack of expansion at ordinary temperatures appears to be merely a belated and not destroyed function. With excessive cold there is likewise a resumption of contraction.

PLATINITE.

Forty-six per cent nickel steel with 0.15 per cent carbon known as platinite, has about the same coefficient of expansion as platinum and glass and for that reason may be imbedded in glass without breaking the latter by a difference in expansion. It has been used in leading wires in the glass bases of electric incandescent lamp bulbs as a substitute for platinum which was formerly held to be indispensable. In those lamp bulbs the preservation of the vacuum is necessary and the joint between the wire and glass must be kept tight. Platinite has not been found wholly suitable for this purpose and is not now so used, a compound wire with a 38 per cent nickel-steel core encased in copper

and sometimes coated with platinum being now generally employed. The nickel-steel core if free will expand less than the glass and the copper more, so that each resists the other and the wire as a whole will have the desired rate of expansion. About 2 tons of nickel steel per year is used in this wire.

Many other alloys of iron and nickel have been studied by Guillet and others.[a] In fact the whole range has received more or less thorough attention, and much knowledge of scientific value has been gained concerning the varieties that so far have not found useful application.

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NICKEL-CHROMIUM STEELS.

Nickel-chromium steels, known in the trade as chrome-nickel steels, are perhaps the most important of the structural alloy steels. Their field of usefulness is continually being enlarged by their application for new purposes and also by encroachment on the premises of some of the other alloy steels, notably of simple nickel steel, and they have almost wholly displaced nickel-vanadium and nickel-chromium-vanadium steels, which several years ago were in some considerable demand.

The amount of nickel-chromium steels produced in 1913 was thought to be about 100,000 tons of ingots, all made in the open-hearth furnace with the exception of 2,000 or 3,000 tons melted in crucibles and electric furnaces. The steel is made by 10 or 12 companies, 2 of which make it at several different plants.

Nickel-chromium steels are seldom used in any but a heat-treated condition. By suitable treatment pieces of small mass can be made to have as high physical properties as any steels known, with any elastic limit between 40,000 and 250,000 pounds per square inch, accompanied by ductility that is high as compared with its strength, as the ductility naturally lessens as the elastic limit increases.

Nickel-chromium steels can be made somewhat more cheaply than simple nickel steel of the same strength and ductility containing a smaller total of the alloying elements, and chromium is less costly than nickel.

COMPOSITION AND PROPERTIES.

The upper limit of nickel in useful chrome-nickel steels is about 3.5 per cent, and all useful steels of this class are pearlitic, according to Guillet.[a] According to the same authority, when a chrome-nickel steel is casehardened, the case is harder than that of a simple nickel steel.

Some of the defects and troubles of chrome-nickel steels are like those of simple nickel steels previously considered.

The composition and properties of six nickel-chromium steels in the natural or untreated state are given in the table following:

Composition and properties of nickel-chromium steels in natural or untreated state.

Sample No.	Composition.							Tensile properties.					Remarks.
	C.	Mn.	Si.	S.	P.	Ni.	Cr.	Tensile strength.	Elastic limit.	Contraction of area.	Elongation in 2 inches.	Ball hardness.	
	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Pounds.	Pounds.	P. ct.	P. ct.		
1	0.55	0.41	0.22	0.03	0.02	1.53	1.14	96,000	75,000	66	31	185	Annealed.
2	.18	.27	.05	.04	.02	1.28	1.59	72,000	51,000	71	37	134	Do.
3	.15	.34	.13	.02	.01	1.28	.37	59,000	42,000	64	38	115	Do.
4	.29	.42	.07	.06	.02	3.86	1.48	....	....	....	....	....	Natural.
5	.25	.32	.10	.03	.02	1.45	1.20	96,509	81,500	68	25	....	Test piece.
6	.25	.32	.10	.03	,02	1.45	1.20	97,100	80,900	49	[a] 7	....	Eyebar; full size.

[a] In 21 feet.

Sample 4 is from a plate similar to that used in the mast of the yacht *Vanitie*. It was not heat-treated but was used as rolled.

Samples 5 and 6 represent the same steel and show the relative properties of the small test piece and the full-size eyebar for a bridge the section of which was 14 by 2 inches. The difference in elongation is particularly noticeable, the great local stretch near the point of rupture being only a small part of the total length of the bar.

The composition and properties of six nickel chromium steels in the heat-treated condition were as follows:

Composition and properties of nickel-chromium steels in heat-treated condition.

Sample No.	Composition.							Tensile properties.					Heat treatment.	
	C.	Mn.	Si.	S.	P.	Ni.	Cr.	Tensility.	Elastic limit.	Contraction of area.	Elongation in 2 inches.	Ball hardness.	Temperature at which steel was quenched in water.	Temperature at which temper was drawn in air.
	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Pounds.	Pounds.	P. ct.	P. ct.		° C.	° C.
1	0.40	0.74	0.24	0.03	0.02	3.45	1.20	187,000	175,000	43	10	352	830	371
2	.36	.53	.11	.04	.01	1.55	.70	145,000	125,000	65	20	233	830	566
3	.21	.41	.22	.03	.02	3.52	1.11	110,000	75,000	66	24	215	830	682
4	.48	.44	.16	.01	.01	2.02	.98	212,000	186,000	46	10	445	843	427
5	.48	.44	.16	.01	.01	2.02	.98	140,000	120,000	61	18	287	843	649
6	.38	.28	.27	.02	.01	3.01	.65	114,000	90,000	69	25	266	843	649

[a] Guillet, Léon, Aciers nickel chrome: Rev. de mét., t. 3, 1906, pp. 462–484.

Any one of the first three samples could be given substantially the properties of either of the other two by varying the temperature of the second heating.

Most of the nickel-chromium steel goes into armor plate, projectiles, and automobile parts.

USE IN AUTOMOBILES.

For automobiles—and the practice might be advantageously extended to other fields—three grades of nickel-chromium steel are used. They are called low, medium, or high according to their contents of nickel and chromium. The carbon content may be varied for each grade within the limits shown in the following table:

Composition of nickel-chromium automobile steels.

Grade.	C.	Mn.	Si.	S.	P.	Ni.	Cr.
Low	0.20 to 0.40	0.65	Low.	0.045	0.04	1.25	0.6
Medium	.20 to .40	.65	Low.	.045	.04	1.75	1.10
High	.20 to .40	.65	Low.	.045	.04	3.50	1.50

These steels are almost invariably heat treated for use in automobiles, a wide range of properties being attainable by varying the heat treatment with each steel. The properties overlap those of steels of both harder and softer grades, so that a wide choice of properties is afforded as well as a choice of steels for the set of properties desired.

USE IN ARMOR PLATE.

An important use for chrome-nickel steel is in both thick and medium armor plate for war vessels. The thick heavy side armor, 6 to 14 inches thick, is face hardened by the well-known methods. A recent analysis of the body of a plate gave: C 0.33 per cent, Mn 0.32 per cent, Si 0.06 per cent, S 0.03 per cent, P 0.014 per cent, Ni 4 per cent, Cr 2 per cent, and its tensile properties after treatment were:

Tensile strength, pounds per square inch	101,000
Elastic limit, pounds per square inch	77,500
Elongation in 2 inches, per cent	24
Contraction of area, per cent	60

The results from such a great mass of metal were excellent.

Medium armor, between 3 to 5 inches thick, is rather similar in composition. It is not face hardened, but is given high properties as a whole by the heat treatment to which it is subjected. An analysis lately made gave: C 0.30 per cent, Mn 0.34 per cent, Si 0.13 per cent, S 0.03 per cent, P 0.03 per cent, Ni 3.66 per cent, Cr 1.45 per cent.

Its physical properties were those given below as sample 1. Sample 2 represented another chrome-nickel steel made for the same purpose, containing 3½ per cent of nickel.

	Sample 1.	Sample 2.
Tensile strength, pounds per square inch	119,000	138,000
Elastic limit, pounds per square inch	106,000	119,000
Elongation in 2 inches, per cent	22	22
Contraction of area, per cent	61	49

Such steel is most excellent for use on warships to form protective decks and barriers to protect from secondary battery fire. Chrome-nickel-vanadium steel is also used for this purpose, as noted elsewhere.

USE IN PROJECTILES AND IN RAILS.

Nickel-chromium steel is used in the manufacture of most armor-piercing projectiles.

Cubillo [a] cites a steel for projectiles, having 0.48 per cent C, 0.58 per cent Mn, 0.75 per cent Cr, 2.55 per cent Ni, 0.40 per cent Si, 0.04 per cent P. A test piece quenched in oil and tempered had an elastic limit of 129,400 pounds per square inch, a tensile strength of 150,300 pounds per square inch, and an elongation of 19 per cent.

For large projectiles Girod [b] prefers chromium-tungsten steel having 0.50 per cent C, 4 per cent Ni, 0 to 1.5 per cent Cr, and 0.25 per cent W.

It is curious that nickel is considered to improve the quality of shot although generally held to injure the quality of high-speed tool steels. In use there seems to be a parallel between the requirements of the two, except for the important and vital difference as to the required speed at which they respectively meet the metal to be penetrated. The speed of impact of the shot enables it to enter when no amount of pressure will effect the same result.

Chrome-nickel steel rails having 2 per cent of nickel and 0.7 per cent of chromium have been tried by several railroads, but with unsatisfactory results. They resisted wear well as compared with simple steel rails, but broke badly both transversely and lengthwise, so that they were considered unsafe and consequently were removed. They were made by the Bessemer process and were not heat treated.

DETAILS OF MANUFACTURE OF A SPECIFIC PIECE OF NICKEL-CHROMIUM STEEL.

Following is a description of the manufacture of a large shaft of mild chrome-nickel steel for marine purposes. A corrugated 35-ton ingot 45 inches in diameter was made of basic open-hearth steel

[a] Cubillo, L., Armor-piercing projectiles: Jour. Iron and Steel Inst., 1913, p. 251.

[b] Girod, P., Discussion of paper on armor-piercing projectiles: Jour. Iron and Steel Inst., 1913, p. 252.

having 0.24 per cent C, 0.70 per cent Mn, 0.013 per cent P, 0.015 per cent S, 0.18 per cent Si, 1.60 per cent Ni, and 0.32 per cent Cr. A few hundredths per cent of titanium was added in the ladle, but did not appear in the steel. The shaft when finished was 14½ inches in diameter, with an 8-inch hole through on the center line.

The steel was melted without the addition of ore late in the heat, a method that favored soundness and tended to allow the steel to clean itself of insoluble impurities such as oxides and silicates. The ingot was forged, annealed at 866° C. (1590° F.), bored, rough-turned, heated to 750° C. (1382° F.), quenched in oil, and drawn at 593° C. (1100° F.).

The shaft was merely raised to the drawing temperature, 593° C., when firing at once ceased, the furnace was closed, and the shaft allowed to cool with the furnace.

The averages of the tests, which were longitudinal, were as follows: Tensile strength, 83,300 pounds per square inch, elastic limit, 52,500 pounds per square inch, elongation in 2 inches 26 per cent, contraction 60 per cent. The results were excellent, though seemingly a lower drawing temperature, which would have resulted in a higher elastic limit, would have been justified.

MAYARI STEEL.

A so-called natural chrome-nickel steel is now made from certain ores mined at Mayari, Cuba. The ores carry enough nickel to give 1.3 to 1.5 per cent of nickel, and enough chromium to give 2½ to 3 per cent of chromium in the crude iron smelted therefrom. When the iron is converted into steel by the pneumatic or open-hearth processes, the nickel is practically all present in the steel, but the chromium is of necessity largely wasted by being oxidized.

Steel made in part of Mayari iron is giving good service in rails and particularly in track bolts, which are heat treated to give the metal an elastic limit of 75,000 pounds per square inch.

Why these rails are satisfactory when other chrome-nickel steels were not has not been explained. The chief differences seem to be (1) that these Mayari steel rails have less of the alloying elements because Mayari iron is used only in part in them, and (2) that the steel is made in the open-hearth furnace.

The use of steel containing Mayari iron is increasing, and the demand is enough to induce the production synthetically of steels of the same composition by adding nickel and chromium to simple steels in the Mayari proportions.

The Mayari steels are not included in the estimated quantity of chrome-nickel steels made, as already given. In fact it is likely that in the near future the tonnage of Mayari steels will surpass that of all the other chrome-nickel steels taken together.

CASTINGS OF NICKEL-CHROMIUM STEELS.

Castings are made also of chrome-nickel steel and may be used in the annealed or heat-treated condition.

Composition and properties of chrome-nickel steel castings.

Sample No.	Composition.							Tensile properties.				Condition.
	C.	Mn.	Si.	S.	P.	Ni.	Cr.	Tensile strength.	Elastic limit.	Contraction of area.	Elongation in 2 inches.	
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Pounds.*	*Pounds.*	*P. ct.*	*P. ct.*	
1	0.30	0.41	......	0.04	0.03	3.64	1.49	91,500	45,500	24	16.5	Annealed.
2	.33	.39	......	.04	.03	3.58	1.61	90,500	46,500	27	18.5	Do.
3	.30	.20	0.35	......	......	2.50	.50	110,000	80,000	30	20	Heat-treated.

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SILICON STEELS.

Although silicon is an ingredient of practically all steels its presence is often accidental or unavoidable, and if it is added to simple structural and tool steels the purpose is to promote soundness rather than to improve the properties of the finished steel. In tool steels silicon is always present, and in times past high-silicon steels have been advocated for tools, but they are not now so used in a commercial way.

MANUFACTURE OF SILICON STEEL.

Silicon steels are generally made in the open-hearth furnace, preferably on an acid hearth, as the acid slag does not waste the silicon in the final additions as rapidly as does a basic slag that contains free oxide of iron, and therefore the final content of silicon desired may be more closely controlled.

Silicon in true silicon steels must be added to the charge only a short time before teeming, as any oxygen that reaches the metal will largely be taken up by the silicon which will be wasted by burning to silicic acid (SiO_2). When so added to a bath in proper condition as to temperature and amount of dissolved oxygen or oxides the silicon will overwhelm the gases in solution, and the steel as cast will be free from blowholes and with a maximum tendency to pipe.

Because of the large proportion of silicon in silicon steels and because of the short time allowable after the silicon has been added to the bath it should be added in the heated or molten state.

Silicon steels containing about 2 per cent of silicon or more roll very "dry," that is, they are liable to be cracked by the heavy reductions of the first passages through the blooming mill.

PROPERTIES OF SILICON STEELS.

Silicon steel containing 0.20 per cent of carbon may be rolled if the silicon content is less than 7 per cent. With 0.90 per cent carbon it may be rolled if the silicon is less than 5 per cent. With a silicon content higher than 5 per cent the metal is useless.[a] In structural steels the effect of the silicon is to raise the elastic limit to a moderate degree. Silicon lowers the coefficient of expansion of steel somewhat as nickel does.

Composition and properties of structural silicon ("silico-manganese") steels.

Sample No.	Description.	C.	Si.	Mn.	S.	P.	Tensile strength.	Elastic limit.	Elongation.	Contraction.	Ball hardness.
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Pounds.*	*Pounds.*	*P. ct.*	*P. ct.*	
1	Automobile springs	0.50	2.0	0.70	0.04	0.03	....	....	....	....	....
2	Springs treated....	.47	1.83	.70	.01	.01	254,000	230,000	9	40	....
3	Carriage axles......	.50	1.90	.70	.04	.04	....	....	....	....	....
4	Test piece, natural condition	.48	1.4	.45	......	......	113,760	71,100	17	....	....
5	Test piece, treated.	.48	1.4	.45	......	......	177,750	149,310	14	....	....
6	Test, treated.......	.50	1.75	.65	......	......	....	198,700	8.5	21	418
7	Annealed..........	.36	1.27	.57	.03	.02	94,500	59,750	25	48	....
8	Drawn at 427° C...	.36	1.27	.57	.03	.02	182,200	160,850	12.5	34	....
9	.....do............	.31	2.39	.48	.03	.05	134,750	104,700	22	55	....

The treated test piece comprising sample 5 was heated to 954° C. (1,750° F.), quenched in water, and drawn at 427° C. (800° F.). The hardening temperature of samples 8 and 9 was probably about the same as that of sample 5.

USES OF SILICON STEELS.

The dividing line between silicon-treated steels and silicon-alloy steels is not clearly defined, but the latter are used for several important purposes. In structural lines their employment is limited, as their properties can, generally speaking, be readily equaled or excelled by simple steels.

The chief structural use of silicon-alloy steel is in springs of the leaf type for automobiles and other vehicles. The silicon is considered to make the springs somewhat tougher so that they are less liable to break in service than springs of simple steel. In the trade this steel is called silico-manganese steel, though its content of manganese is

[a] Guillet, Léon, Aciers au silicium: Rev. mét., t. 1, 1904, p. 67.

usually no more than is common in simple steels and not enough to properly cause the steel to be classified as a manganese-alloy steel.

In electricity, an important use for silicon-alloy steel is in the cores of static transformers. With the exception of manganese most of the elements employed in making alloy steels, although not greatly decreasing the magnetic susceptibility of the iron that contains them, lower its hysteresis loss. Silicon is the element most used for that purpose because it is the cheapest, but aluminum, phosphorus, nickel (3½ per cent), and tungsten have a similar effect.

The silicon content in silicon transformer metal is usually between 3½ and 4½ per cent or, more exactly, 4 to 4½ per cent. Some 25,000 tons was used in 1913 for this purpose. The steel is rolled into thin sheets which, for one large user, are 0.014 inch thick; the transformer cores are built up of these sheets, which are cut to shape separately by stamping. For low induction the permeability of this steel is nearly as great if not greater than that of any other variety of iron or iron alloy known, and its hysteresis loss is less than that of any other of nearly as low cost.

The results of an analysis of a transformer core made of silicon-alloy steel are as follows: C, 0.08 per cent; Si, 4.18 per cent; Mn, 0.11 per cent; S, 0.06 per cent; P, 0.01 per cent; Al, 0.01 per cent.

Silicon steels can not be case hardened as the silicon retards the absorption of carbon; the silicon content must therefore be low, not over 0.04 per cent, in steel intended to be so treated.

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HIGH-SPEED TOOL STEELS.

High-speed tool steels, also called rapid steels, have in the past fifteen years worked a remarkable revolution in the machine-shop business of the whole world, affording largely increased outputs and commensurate lower costs. As a consequence they are now being used very generally and in some shops almost exclusively for machining iron and steel as well as some other metals by cutting operations by machine tools.

The revolutionary feature wherein tools made of these steels differ from and exceed in service the tools formerly used is their ability to

maintain a sharp strong cutting edge while heated to a temperature far above that which would at once destroy the cutting ability of a simple steel tool. Because of this property a tool made of high-speed tool steel can be made to cut continuously at speeds three to five times as great as that practicable with other tools, and when, as the result of the friction of the chip on the tool, it may be red-hot at the point on top where the chip rubs hardest, and the chip itself may, by its friction on the tool and the internal work done on it by upsetting it, be heated to a blue heat of 296° C. (565° F.) or even hotter to perhaps 340° C. (644° F.).

This property of red-hardness or ability to retain hardness at a red heat may be imparted to steels of suitable composition, comprising chromium and tungsten, by a unique heat treatment to which they may be subjected. This treatment, described later, was introduced by F. W. Taylor and Maunsel White, as has been described by Taylor,[a] at the works of the Bethlehem Steel Co. in 1899, and tools so treated were shown at the Paris Exposition in 1900, where they naturally created a great sensation among those familiar with the machining of metals. White, when giving the writer his first knowledge of these tools in 1899 or early in 1900, said that a young man in the Bethlehem shop had lighted a cigarette with a newly cut chip, a statement that seemed almost unbelievable at the time.

In this country in 1913 about 7,000 tons of high-speed or rapid tool steels was made by some 15 makers, that output requiring about 8,000 tons of ingots.

MANUFACTURE OF HIGH-SPEED TOOL STEEL.

High-speed tool steels are all made by the crucible or electric-furnace process. Except at one works, the crucibles or pots are made of graphite. The average life of the crucibles or pots varies in different works from six to nine melts. Some makers use clay-lined graphite pots in melting this steel to prevent or hinder the absorption of carbon from the pot. The clay lining is only one-eighth to three-sixteenths of an inch thick and is sometimes cut through on the second or third melt; in that event the molten steel may absorb too much carbon. Other makers use a graphite pot twice—first for melting other kinds of steel and then for rapid steel when the inner surface of the pot is somewhat slagged over, because of which the absorption of carbon is much less than when the pot was new.

The large producers use gas-fired melting furnaces for heating the pots, which are charged into the furnace at the top. Each melting hole contains six pots and each pot takes a charge of 90 or 100 pounds.

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[a] Taylor, F. W., On the art of cutting tools: Trans. Am. Soc. Mech. Eng., vol. 28, 1906, pp. 31-350.

The charge is melted and then "killed" in the usual way by being held 30 to 40 minutes. Such procedure, together with the presence of the large amount of alloy, regularly gives sound piping steel. If run continuously a furnaceful of pots will be melted about every four hours.

In packing a pot with a charge for rapid steel the tungsten must be placed on top of the charge—as with simple tungsten steel—to guard as far as possible against the tendency of the tungsten to settle because of its high specific gravity. That tendency seems to be less with the rapid steels than with the simple tungsten steels. Whether the chromium of the former influences or hinders the settling of the tungsten is conjectural.

The smaller ingots, which are made from one pot of steel, vary from 3½ to 5 inches square. The steel is sometimes teemed directly into the mold by hand-pouring, but in some works clay funnels are placed on top of the mold to direct the stream down the center of the mold to avoid cutting its wall, as might happen if the stream impinged on it. Funnel pouring is also advantageous when two pots are to be combined to make a larger ingot, as the steel can be poured into the funnel from opposite sides at the same time, a procedure that will mix the liquid steel and give a more uniform ingot than when one pot follows another, as in hand pouring when no funnel is used.

Some of the larger producers of rapid steels use for casting a large bottom-pouring ladle into which the steel is poured from the pots of one or more furnaces, and from which the ingots are top-cast; that is, the molds are filled from the top. This method presents the advantages that (1) the product is more uniform; (2) individual pot charges, which might not be of the prescribed composition or might be otherwise unsatisfactory, are merged with the others without detriment to the whole; (3) large ingots are easily made; (4) one analysis serves for the whole number of pots; (5) one test serves for the whole ladleful of steel. It is a matter of experience that complaints from customers became much less frequent after the introduction of the ladle for casting this steel.

The strong tendency of rapid steel to pipe is checked considerably in most plants by the use on each ingot of a hot "dozzler," which is a clay ring preheated red hot, that is placed on the ingot top and filled with molten steel. This arrangement keeps the top of the ingot molten long enough so that the pipe is of diminished size and nearly or quite all contained within the part of the ingot surrounded by the dozzler. The proportion of the ingot to be rejected on account of the pipe is therefore much decreased. The molds are usually closed at the bottom end and are either made with parallel walls or tapered so that the ingot is larger at the top than at the bottom. The molds

must be split when the walls are parallel, and are sometimes split when the ingots are tapered.

High-speed tool steel as cast has a coarse structure and dark color as compared with the structure and color of simple steels of the same carbon content. A corner is broken from the top of each ingot, to show the grain, and the ingots when hand-poured directly from the pots are classified by the eye as in the production of simple crucible steels. If the ingots are cast from the large ladle a test is taken for analysis which determines the disposition of the whole ladleful of steel.

As a rule the ingots show a strong columnar structure or arrangement of crystals whose axes are normal to the cooling surface. Some makers refer to the structure as a "lemon structure," the crystals of the metal being thought to resemble the cells forming the pulp of a lemon. If the casting temperature is lower than usual, this lemon structure may be absent, and in that case the interior of the ingot will have a much finer grain than the ingots cast at the usual higher temperature. The subsequent heating and working of the steel entirely destroys the crystalline structure of the ingot, and the worked steel, on a fresh fracture, shows a most beautiful porcelanic structure.

The ingots run from 3½ by 3½ inches to 16 by 16 inches, but most of them are from 5 by 5 inches to 9 by 9 inches. For hot-working they are heated in a furnace chamber having a temperature of about 1,180° C. (2,156° F.). At this high heat the steel may be worked satisfactorily under the hammer or press and may be quickly worked down to the dimension desired.

COMPOSITION OF HIGH-SPEED TOOL STEELS.

The tendency of the makers is toward a somewhat uniform composition as regards the contents of the alloying elements, whose benefits have become fairly well known, and whose use as a consequence may be considered as established. Specifically, these alloying elements are tungsten and chromium. The addition of vanadium and cobalt in important proportions is considered by some makers to give distinct improvement to high-speed steel, and some vanadium is almost always present.

The following analyses are of steels recently made, most of which are considered to be good commercial steels:

Results of analyses of high-speed steels made in 1913 or 1914.

Sample.[a]	C.	Mn.	Si.	S.	P.	Cr.	W.	V.	Co.	Ni.	Mo.	Remarks.
	P.ct.	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	
A	0.65	0.15	0.20	0.02	0.03	4.75	17.50	0.90	......	......	......	
B—1	.66	.27	.14	.04	.05	4.51	17.48	.70	4.22	0.17	......	
B—2	.74	.31	.13	.04	.02	4.20	15.63	.67	2.70	......	......	
B—3	.63	.14	.07	.04	.05	4.26	17.16	.45	3.80	.20	......	
B—4	.69	.34	.14	.03	.04	5.28	16.35	.64	5.28	......	......	
C—1	.66	.22	.17	.03	.02	3.44	16.51	.73	......	......	......	
C—2	.64	.21	.16	.03	.03	3.30	16.06	.66	4.02	......	......	
C—3	.67	.33	.25	.02	.02	3.85	16.06	.70	......	......	......	
D—1	.75	.28	.36	.03	......	4.10	19.00	.75	......	......	......	Good.
D—2	.68	.38	.40	.03	......	4.65	17.85	.53	......	......	......	Inferior.
D—3	.69	.36	.38	.04	......	4.67	17.90	.50	......	......	......	Do.
D—4	.57	.20	.26	.02	.03	4.82	15.38	.50	......	......	......	Do.
E—1	.61	.23	.35	.04	......	4.10	17.20	1.00	......	......	......	Good.
E—2	.68	.45	.40	.04	......	4.00	14.26	1.09	......	......	......	Inferior.
E—3	.70	.50	.39	.05	......	4.08	14.50	1.07	......	......	......	Do.
E—4	.60	.23	.12	.03	.02	3.90	17.27	.90	......	......	......	Do.
F	.64	2.29	.12	.02	.01	4.39	16.09	.59	......	.28	......	
G	.72	.37	.18	.03	.02	4.50	13.30	2.50	......	......	......	
H—1	.77	.16	.21	.02	.02	4.05	18.64	1.35	......	......	......	
H—2	.67	.16	.20	.02	.02	4.66	13.86	1.08	......	......	......	
I	.64	.23	.29	.02	.02	4.57	19.10	.54	......	......	......	
J—1	.64	.30	.26	.02	.01	2.93	18.71	1.22	......	......	......	
J—2	.71	.14	.26	.03	.03	2.97	18.21	.97	......	......	......	
K—1	.55	Tr.	.23	.02	.04	4.46	16.05	.80	4.72	......	0.72	
K—2	.70	Tr.	.18	.01	.02	4.25	15.50	.88	4.72	.18	.67	
K—3	.74	.31	.13	.04	.02	4.20	15.63	.67	2.70	......	......	

[a] Samples A to I represented American steels, the numerals indicating different samples from the same maker; sample J represented an English steel; sample K represented a German steel.

Samples D—1 and E—1 gave excellent results in a competitive test, whereas samples D—2, D—3, E—2, and E—3, manufactured by the same makers, gave distinctly inferior results in the same shop.

The occurrence of nickel in four of the samples may have been accidental, having been due to nickel in some of the scrap steel used in the charge. Most makers now put in vanadium, and steel like that represented by sample G, which had the highest vanadium content of all the samples represented in the table, was the winner in a recent competitive test.

The average specific gravity of the steels represented in the table was about 8.8, the increase over the specific gravity of iron being due chiefly to the tungsten content.

There are so many factors beside the ultimate composition that affect the value of rapid tool steels that no conclusion can be drawn from the analysis alone. The melting, hot working, and heat treatment all must be done correctly or the final result will not conform to expectations.

CARBON IN HIGH-SPEED TOOL STEEL.

The proportion of carbon aimed at in high-speed tool steels is about 0.65 per cent, which in a simple steel would not be enough to give the maximum hardness even if the steel were heated above the critical point and quenched in water, and still less so when the steel is cooled as slowly as these steels are in their treatment. This shows that the carbon acts in a different way from what it does in simple steels, as is discussed later.

TUNGSTEN IN HIGH-SPEED TOOL STEEL.

Tungsten is well established as a most important if not indispensable ingredient of commercial tool steels, being almost or quite universally used in quantity therein. The best proportion of tungsten, all things considered, seems to lie between 16 and 20 per cent, the tungsten content in 95 per cent of all the American steels coming within these limits. Some published analyses of European high-speed tool steels show a higher content of tungsten than this, but American makers generally agree that any tungsten in excess of 20 per cent adds nothing to the usefulness of the steel, and they therefore make that proportion the upper limit of the amount added. One effect of the tungsten is that the best percentage of carbon in rapid steel is but about half that required in simple tool steels intended for the same kind of service.

CHROMIUM IN HIGH-SPEED TOOL STEEL.

The effect of chromium in high-speed tool steel, as in other steels, is undoubtedly as a hardener, entering into the double carbide of tungsten and chromium which gives or causes the proper cutting edge. Although the proportion of this element present in these steels varies considerably, it is always large, perhaps never less than 2 per cent or more than 6 per cent in American steels, and in European steels the upper limits is at least 9 per cent.

MOLYBDENUM IN HIGH-SPEED TOOL STEEL.

The use of molybdenum in high-speed tool steels is being generally discontinued. Some makers for years manufactured molybdenum tool steels, but as a rule they have either wholly discontinued its use or use a much smaller proportion than formerly, employing it as an auxiliary rather than a major constituent.

The effect of molybdenum is similar to that of tungsten, but is more intense in that 1 per cent molybdenum is currently considered to give about the same or greater hardening effect than 2 per cent of tungsten. It gives a fine cutting edge.

Various reasons are assigned for the discontinuance of the use of molybdenum in these steels. Taylor[a] found that molybdenum in rapid steels caused irregular performance; that steels of nearly the same composition and having had seemingly the same treatment gave large variations in the cutting speeds they would stand. One user specifies no molybdenum because it causes the tools to crack in quenching. A maker objected to molybdenum because molybdenum steel was apt to be seamy and to contain physical imper-.

[a] Taylor, F. W., On the art of cutting metals: Trans. Am. Chem. Soc., vol. 28, 1906, pp. 31-350.

fections. A maker of ferro-alloys understands that molybdenum steel deteriorated upon repeated heating for dressing and treatment, seemingly because the molybdenum disappeared from the outer parts of the steel where it was exposed to the heat and where any volatile constituent could escape. This phenomenon recalls a proposal of Moissan to use molybdenum as a means of freeing molten iron of oxygen, as he found that the oxide of molybdenum was much more volatile than the metal itself. It may be, therefore, that the molybdenum at the surface is oxidized and volatilized, but for this waste to extend to any considerable depth implies either that molybdenum moves through the heated steel to reach the oxygen of the air, or, as is more likely, that the oxygen of the air penetrates the steel, reaching the molybdenum and oxidizing it, the volatile oxide escaping from the steel.

The cost of molybdenum when its use was most general was much more than that of tungsten. Because of the falling off of the demand in recent years, however, the price of molybdenum has fallen to about that of tungsten, or to a fraction of what it was a few years ago.

It has been observed that in the Eggertz color determination a high molybdenum content (4 to 8 per cent) causes "missing" or invisible carbon when the content of latter element is high (0.4 to 1.2 per cent).

VANADIUM IN HIGH-SPEED TOOL STEEL.

Vanadium is used for high-speed tool steel in varying amounts, most makers using at least 0.5 per cent, although some run the vanadium content up to 1½ or 1¾ per cent, or even more, considering that such an addition increases in an important degree the value of the steel for tools.

The effect of vanadium is considered to resemble in some ways that of chromium in increasing the hardness or red-hardness of the cutting edge. One maker makes two kinds of high-speed tool steel, the essential difference between the two being that one contains about 1½ per cent of vanadium and thereby commands a higher price, whereas the other has little of this element, both kinds being intended for substantially the same service.

High-speed steels containing vanadium are generally classed as "superior" steels and many though not all makers and users consider them distinctly better than the "standard" steels containing no vanadium, both on account of their actual cutting qualities at high speeds and on account of the length of time a tool will cut before it needs regrinding. The true value of vanadium in rapid steels must probably be held as not yet fully determined. The

analyses given in the table show that all the samples contained vanadium but in greatly varying amounts.

COBALT IN HIGH-SPEED TOOL STEEL.

Cobalt now threatens to change tool-steel manufacture because of the properties it imparts. The recent great decline in price following the increase of the supply from the silver ores of the Cobalt district in Ontario naturally led to its trial as a steel-alloying element, and some most excellent high-speed steels containing, in addition to the usual ingredients, about 4 per cent of cobalt, have been obtained. This result was hardly to have been expected in view of the experience with nickel, which cobalt much resembles, as nickel has been condemned by nearly every manufacturer as not being a desirable ingredient of high-speed tool steels, because of the effect it has of making the edge soft or "leady." The cobalt steel, however, has shown, in some products at least, increased ability to hold its edge in work.

One user of cobalt steel found it better suited for turning manganese steel than any other steel he tried, his success being so marked as to make it practically a commercial operation. Manganese steel, as noted elsewhere, is so hard as to be considered practically unmachinable, the usual practice having been to finish it by grinding when necessary. Another user found that an imported cobalt high-speed steel proved excellent for cutting a hard nickel-chromium steel in a lathe, whereas the same steel in a cold saw was not satisfactory. The direct reason seemed to be that this steel to do its best work must be run nearly if not quite red hot; at least that was the condition in which it was used in the lathe, while in the cold saw its temperature was not raised to an important degree, and when cold its properties were different than when it was hot. This behavior is more or less common to high-speed steel as is mentioned later.

The valuable effect of cobalt is claimed to be that it increases the red-hardness of high-speed tool steel, enabling the steel to cut at a higher speed.

COPPER IN HIGH-SPEED TOOL STEEL.

Copper has been considered to be highly injurious in high-speed tool steel, even as little as 0.05 per cent being inadmissible; and it is thought to be particularly harmful if much sulphur is present in the steel; also the higher the carbon content the more harmful is the copper.

SULPHUR AND PHOSPHORUS IN HIGH-SPEED TOOL STEEL.

Sulphur and phosphorus, which are so deleterious in simple tool steels, are considered to be somewhat less so in high-speed steels, in which the effect is either modified or else masked by the large quantities of other ingredients. Some commercial brands of high-speed steels have as much as 0.05 per cent of each of these impurities, to which no inferior quality is attributable.

STELLITE.

Stellite, though a competitor of high-speed steels, is not within the scope of our subject, but a recent analysis is given of a sample for such interest as it may have in relation to cutting steels.

Analysis of stellite.

Constituent.	Per cent.
Co	59.50
Cr	10.77
Mo	22.50
C	.87
Si	.77
Mn	2.04
S	.084
P	.040
Fe	3.11
W	0
Ni	0
	99.684

HEAT TREATMENT OF HIGH-SPEED TOOLS.

The heat treatment given to high-speed steels for the commoner uses as lathe and planer tools has generally been simplified to heating to incipient fusion and quenching in oil. Cooling by an air blast and double treatment, which were formerly recommended, are now not common, except that a second (drawing) heating is given to milling cutters and similar tools, the temperature imparted to the tool depending on the material to be cut.

The treatment is usually done by the blacksmith who heats the tool in his forge fire and then immerses it in a tank containing enough oil so that its temperature does not rise materially. Ten gallons of oil is a common quantity to use when the size and number of the tools is moderate, as in most shops. The fire is a deep compact coal fire, the coal in the center where the tool is heated being pretty thoroughly coked, that is, most of its volatile matter distilled out. This manner of heating has the advantage that free oxygen does not get at the tool to oxidize it, but its environment is nonoxidizing or even reducing, owing to the presence of an excess of burning carbon

surrounding the tool. Any flame is more or less oxidizing, at least unless heavily charged with smoke or free carbon, and a piece of steel heated directly by a flame as in the ordinary heating chamber of a furnace is likely to be somewhat oxidized on its surface, the depth to which the oxygen penetrates varying according to the conditions, particularly the temperature, the access of air, and the length of time. Heating in a muffle will also result in oxidizing the steel unless extraordinary precautions are taken to keep out oxygen or to consume all that enters. The temperature of quenching, usually about 1,260° C. (2,300° F.), is determined by the fusion of the scale and its visible collection into drops or beads on the surface of the tool.

Quenching is done by quickly plunging the heated tool into the oil as soon as it has reached the desired temperature and moving it about in the oil until cold. Cooling in oil is thought by some to give a better tool than cooling in the air blast, one reason seemingly being the protection of the steel from free oxygen while it is hot enough to be oxidized thereby. The oxygen of the air blast forms a scale of oxide on the hot steel and the oxygen probably penetrates the metal below the scale to some extent, injuring the quality as deep as it goes. A tool on its second grinding when the oxidized metal is removed may then give better service than on the first, unless the first grinding has for that reason been heavy enough to remove the oxidized metal.

In some shops, however, the original treatment recommended by Taylor and White [a] is given, the cutting edge of the tool being heated to incipient fusion and then immersed in a bath of melted lead at about 565° C. (1050° F.). The heating is done in a small furnace over a deep coke fire, blown by an air blast, so that the environment of the tool while being heated is substantially nonoxidizing. Flames of carbonic oxide play out of the openings through which the tools are inserted, indicating little if any free oxygen within. In these shops, however, milling cutters and other tools that are machined to a particular form are treated by heating them to a slightly lower temperature, in order not to damage the cutting edges, and then plunging them into cold oil.

When cooled to the temperature of the lead it is taken out and placed in an air blast to complete the cooling. Some tools desired to be especially tough so as not to break in service are given a second heating to 565° C. and then cooled in the open air or air blast if saving time is important.

Rapid steel when well annealed will bend considerably without breaking, even in as large a section as 2½ by 1¼ inches, the bending being edgewise, as in a tool at work.

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[a] Taylor, F. W., On the art of cutting metals: Trans. Am. Soc. Mech. Eng., vol. 28, 1906, p. 228.

Gledhill[a] found that one of these steels after having been annealed 12 to 18 hours at 760° C. (1400° F.) had a tensility of 129,200 pounds per square inch, an elastic limit of 89,600 pounds per square inch, an elongation of 18 per cent in 2 inches, and a contraction of area of 35 per cent. The ductility is rather high and would enable a tool to be bent considerably without breaking. Such annealed steel may be rather easily machined for making milling cutters and other shapes that require machining.

Carpenter[b] found that the higher the temperature from which rapid steel is cooled the more it resisted etching for metallographic work. He also found that no tempering change occurred when it was reheated at a temperature of less than 550° C. (1022° F.) to a visible red in the dark, indicating a stability that is doubtless the cause of its property of red hardness.

Whether a rapid steel is made harder by the heat treatment given it depends somewhat on the condition of the bar before treatment. If it has previously been annealed, the treatment hardens it, whereas heat treatment may not harden a piece in the natural state. Taylor[c] found that some tools having useful red hardness could be filed rather readily. Edwards[d] on the other hand found treated high-speed steels to be exceedingly hard—as hard as any steel could be made by quenching. Gledhill[e] found that high-speed steel was good for turning chilled rolls which are extremely hard and require to cut them the hardest kind of tool.

Trials on window glass of a number of different rapid steels showed that the cutting edge of some but not of all would scratch it. The same was true of the untreated ends of the same tools, as some would and some would not scratch the window pane.

The hardness of the steel when cold is not the determining factor of usefulness in any case. It is the hardness when heated under conditions of work.

The cutting edge of a rapid-steel tool at work is probably never as hot as the metal just back of it, where the heating caused by the friction of the chip, as it is deflected and rubs hard on the tool, is most intense. The edge itself is kept relatively cool by the cold metal flowing upon it.

[a] Gledhill, J. M., The development and use of high-speed tool steel: Jour. Iron and Steel Inst., pt. 2, 1904, pp. 127-181.

[b] Carpenter, H. C. H., The types of structure and the critical ranges on heating and cooling of high-speed tool steels under varying thermal treatment: Jour. Iron and Steel Inst., pt. 1, 1905, pp. 433-473.

[c] Taylor, F. W., On the art of cutting metals: Trans. Am. Soc. Mech. Eng., vol. 28, 1906, pp. 31-350.

[d] Edwards, C. A., Function of chromium and tungsten in high-speed tool steel: Jour. Iron and Steel Inst, pt. 2, 1908, pp. 104-132.

[e] Gledhill, J. M., loc. cit.

THEORY OF HIGH-SPEED STEELS.

Carpenter[a] found the heating and cooling curves of a rapid steel to be radically different from each other, and also that the cooling curve when the steel was cooled from 930° C. (1706° F.) was greatly different from that when the steel was cooled from 1250° C. (2282° F.). When the steel was cooled from 930° C. the curve had an abrupt jog, which showed a great retardation in rate of cooling occurring between 700° C. and 750° C. (1292° F. to 1382° F.). The jog did not occur when the steel was cooled from 1250° C., 320° higher, the line representing variations in rate of cooling being nearly straight. The rate of cooling to get these curves was slow or at least not accelerated, and one can not say what the curves would be like if the rate of cooling were hastened, as in quenching, but the curves obtained seem to throw much light on the question. The property of red hardness seems to be connected with the elimination of the great retardation mentioned.

The following explanation, based on the work of Carpenter[a] and Edwards[b], of the properties of high-speed steels, seems to be helpful or even satisfactory:

Their researches on the heating and cooling of these steels have shown that such steels have an extraordinary stability of composition after they have been heated to 1,200° C. (2,192° F.) or more, and that a second heating of 550° C. (1,022° F.) has no softening or drawing effect. It seems fairly evident that red-hardness depends on or is the natural result of these facts.

At a temperature higher than 1,200° C. (2,192° F.) a double carbide of chromium and tungsten is formed, which persists largely even when the steel is cooled slowly as in the open air, and more so when cooling is accelerated. This double carbide imparts to the steel a high degree of hardness and is stable at all temperatures up to 550° C. (1,022° F.) or somewhat higher. At 550° C. the steel has a low red color visible in the dark.

If the above theory be true then at a temperature of 1,200° C. (2,192° F.) the chromium and tungsten must have a stronger affinity for carbon than iron has, whereas at lower temperatures, say from around 930° C. down to the critical point, the affinity of carbon for iron is slightly stronger than that of either chromium or tungsten or both, and the carbon then exists wholly or in part as carbide of iron, or a complex carbide of iron with one or both of the other elements.

Carbide of iron, or hardening carbon which causes the hard condition of iron in simple steel that has been quenched from a temperature higher than the critical point, is unstable at even slight elevations of temperature above atmospheric temperature, its unstable-

[a] Carpenter, H. C. H., loc. cit. [b] Edwards, C. A., loc. cit.

ness increasing with the degree of heat, though not being proportional thereto. Boynton[a] has shown that between 400° C (752° F.) and 500° C. (952° F.) the amount of change and consequent softening is much greater than at other temperatures, either lower or higher.

The proportion of carbon in rapid steel should perhaps be only as much as will combine with the chromium and tungsten at 1,200° C. (2,192° F.) and leave none to exist as unstable hardening carbon of hardened simple steel.

TESTING AND USING HIGH-SPEED STEEL.

A reliable and inexpensive method of quickly testing high-speed steels to show their value is much needed, as Taylor[b] has explained. Herbert[c] and Edwards[d] have used and recommended machines and methods that lessen the time and trouble of testing, but no test seems to take the place of a trial at actual work because the performance of a tool in one line of work with certain conditions may not be foretold positively by its performance in another with different conditions. Among the reasons are that (1) sometimes greater durability is obtained by changing, that is, increasing or lessening, the speed of the cut, thus changing also the temperature of the tool or (2) a given tool when used at its best speed may be excellent for cutting a certain material, yet prove inferior to another tool for cutting a different material. Thus if selected as the best by trial for cutting a 0.20 per cent carbon steel it may be surpassed by others in cutting a 0.70 per cent carbon steel.

Physical tests of rapid steels at different temperatures up to 800° C. (1,472° F.) are needed to show the effect of heat on the physical properties of those steels. New uses would probably be suggested by the results of such a series of tests.

A rapid-steel tool does not finish the piece being cut as nicely as does a simple steel tool, as the rapid steel does not keep a fine edge with a light cut and slow speed of, say, 20 feet per minute. The durability of such a tool taking a light cut is much greater at a higher cutting speed, at which the tool is hotter, showing that the strength or the toughness of the steel or both are augmented by the higher temperature. Unhardened simple steels with 0.6 to 0.7 per cent carbon get stronger but less ductile with a rise of temperature up to about 300° C. (572° F.). If, as the temperature rises, high-speed steels get stronger without loss of ductility but perhaps with an in-

[a] Boynton, H. C., Hardness of the constituents of iron and steel: Jour. Iron and Steel Inst., 1906, p. 287.
[b] Taylor, F. W., On the art of cutting metals: Trans. Am. Mech. Eng., vol. 28, 1906, pp. 31-350.
[c] Herbert, E. G., The testing of files and tool steels: Trans. Manchester Assn., Eng., 1908-1909, pp. 302-317.
[d] Edwards, E. T., Composition of high-speed tool steel: Iron Age, vol. 89, April, 1912, pp. 957-960.

crease, within limits of course, a physical reason for their great durability is provided.

In 1910 Herbert[a] announced the discovery that any rapid-steel tool and some simple steel tools may have two rather widely separated cutting speeds at which the tool is more durable than at speeds above, below, or between. Thus out of many cases described, one tool cooled in an air jet had nearly equal maximum durability at two speeds—50 and 90 feet per minute, whereas at 65 feet the durability was less than one-half of that at either of the other speeds. This discovery no doubt accounts for some of the anomalies encountered in tool steels as well as other steels the properties or performances of which are not what would be expected from their composition and other attributes. Thus a tool may be condemned when an increase of its cutting speed would cause it to give satisfactory service and durability.

Rapid steel will do its best cutting when hot. A desirable practice followed in some shops, is to heat a tool to near redness before putting it to work.

MACHINE-TOOL DESIGN.

When Taylor and White first introduced rapid steels it was thought that the higher cutting speeds afforded constituted the sole benefit to be derived from them, and as the higher speeds, although consuming more power about in proportion to the increase in speed, did not increase materially the stresses on the machine tools, it was thought that the latter merely needed to be speeded up in order to get the full benefit from the new steels. But it was soon found that the rapid steels in addition to cutting at higher speeds were capable of taking much heavier cuts, which proportionately increased the stresses on the tools. To take full advantage of the heavier as well as the more rapid cuts, machine tools were generally redesigned to provide the greater strength required, and were supplied with proportionately more power. The resulting economies all over the world have been enormous.

The advantages from the use of rapid steels as compared with the use of simple tool steels are in the lessened costs of the ordinary operations of finishing iron and steel because of:

1. More rapid cutting speed.
2. Heavier chips cut, hence larger cuts and feeds.
3. Saving of power per unit of metal cut off.
4. Lower cost of plant for a given output.
5. Lower general and overhead charges connected with manufacturing iron and steel products.

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[a] Herbert, E. G., The cutting properties of tool steel: Jour. Iron and Steel Inst., 1910, pt. 1, p. 216.

PATENTS ON HIGH-SPEED STEELS.

Since the original Taylor and White patents for treatment of high-speed steels were issued in 1901, others have been granted for almost every possible combination of elements which were in any way thought to be useful or valuable constituents of tool steel. Chromium and tungsten were generally included, though not always. Nevertheless most makers now aim at substantially the same foundation composition, with varying amounts of vanadium and sometimes with cobalt.

MISCELLANEOUS USES OF HIGH-SPEED STEELS.

An important use for high-speed steel is in the exhaust valves for automobile engines, where it has given excellent results. These valves operate sometimes at a red heat and seemingly the property of red-hardness that the steel possesses enables it to give good service in these valves.

High-speed steel is being used also in the manufacture of extruded brass to form the die through which the extruded metal is forced. The temperature of the brass is high, near its fusion point, and seemingly the red-hardness of the steel enables the steel to perform this service satisfactorily.

A good file or a good cold chisel may be made of rapid steel, but they are not good enough to justify their cost as compared with those made of simple steels.

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CHROMIUM-VANADIUM STEELS.

Chromium-vanadium steels, usually called in the trade chrome-vanadium steels, are the latest development in structural alloy steels that have gained an extensive market. In 1913 about 90,000 tons of ingots is thought to have been made, of which about 75,000 tons

was sold in rolled and forged products. These steels are almost all made in the open-hearth furnace, the chromium and vanadium alloys being added shortly before casting.

The hot working of chrome-vanadium steels presents no especial difficulties. The total amount of alloying elements is not large in the commercial grades, and the steel acts in the press and rolls much like simple steels with somewhat higher carbon contents.

Chrome-vanadium steels are in their physical properties much like chrome-nickel steels, but they have a greater contraction of area for a given elastic limit than the latter.

This higher contraction of area in the pulling test seems in some way to be associated with machinability, as chrome-vanadium steel with an elastic limit of 150,000 pounds per square inch may be machined rapidly, whereas a chrome-nickel steel having such an elastic limit would quickly dull the cutting tool if cut at the same speed.

The greater part of the chrome-vanadium steels made goes into automobiles. They are preferred by some because of their greater freedom from surface imperfections, notably seams, which steels containing nickel are prone to have if the ingots are at all unsound. Vanadium is a deoxidizer, whereas nickel is not, so that vanadium, when present, favors quality, and the smaller proportion required enables it to compete with nickel even though its cost is five or six times as great.

Chrome-vanadium steels are nearly always used in the heat-treated condition, but there are exceptions even in automobiles, as some frames, forgings, and shafts are made of the steel in its natural state. When heat-treated these steels are both hardened and drawn at slightly higher temperatures than are used with nickel-chromium steels to get similar properties. These temperatures are given in the table of heat-treated chrome-vanadium steels.

Some chrome-vanadium steel is said to be used in armor plate of medium thickness (4 inches), which is not face-hardened but has high properties imparted by heat treatment. Some such steel is used in high-duty forgings and structural parts of machines.

Composition and properties of chrome-vanadium steels in natural state.

Sample No.	Composition.							Tensile properties.				
	C.	Mn.	Si.	S.	P.	V.	Cr.	Tensile strength.	Elastic limit.	Contraction of area.	Elongation in 2 inches.	Ball hardness.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Pounds.*	*Pounds.*	*Per cent.*	*Per cent.*	
1	0.57	0.84	0.27	0.03	0.01	0.31	1.36	98,000	75,750	68.5	28.1	175
2	.46	.48	.20	.02	.01	.14	1.17	82,250	52,500	71.0	34.0	160
3	.18	.32	.18	.02	.01	.20	.74	60,500	42,900	75.0	43.0	133
4	.30	.65	.10	.04	.04	.18	.90	....	45,000	69.0	35.0	[a] 155

[a] Annealed.

Composition and properties of chrome-vanadium steels in heat-treated state.

Sample No.	Composition. C.	Mn.	Si	S.	P.	Cr.	V.	Tensile properties. Tensile strength.	Elastic limit.	Contraction of area.	Elongation in 2 inches.	Ball hardness.	Treatment.[a]
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Pounds.*	[illegible]	[illegible]	*P. ct.*		
1	0.30	0.65	0.10	0.04	0.04	0.90	0.18	....	[illegible]	[illegible]	20	255	899° W; 704° A
2	.30	.65	.10	.04	.04	.90	[illegible]	....	[illegible]	[illegible]	10	480	899° W; 454° A.
3	.30	.65	.10	.04	.04	.90	.18	....	[illegible]	[illegible]	10	429	899° W; 315° A.
4	.28	.45	.26	.02	.01	1.00	.18	96,500	[illegible]	[illegible]	34	187	899° O; 676° A
5	.40	.75	.26	.01	.01	1.00	.17	148,000	[illegible]	[illegible]	20	270	926° O; 676° A.
6	.40	.75	.26	.01	.01	1.00	.17	221,000	[illegible]	[illegible]	11	435	926° O; 426° A.
7	.67	.37	.20	.02	.01	.89	.22	188,200	[illegible]	[illegible]	14	330	——; 426° A.
8	1.06	.36	.22	.02	.02	.95	.11	135,550	[illegible]	[illegible]	21	[illegible]	——; 648° A.
9	.41	.49	.12	.03	.03	1.09	.11	86,900	[illegible]	[illegible]	33	[illegible]	——; 754° A.
10	.25	.50	.10	.08	.02	.95	.75	131,700	[illegible]	[illegible]	18	......	....

[a] The first temperature given for each sample is that at which the steel was quenched, and the second the drawing temperature; W, O, and A represent water, oil, and air, the three cooling media used. Samples 8, 9, and 10 were hardened before being drawn at the temperatures given.

EXAMPLE OF SATISFACTORY USE OF CHROME-VANADIUM STEEL.

A hydroelectric plant had shafts 6½ inches in diameter, which transmitted 3,000 kw. each at 480 revolutions per minute, and all broke in service. The shafts were made of untreated nickel steel having an elastic limit of about 40,000 pounds per square inch. To make stronger shafts by increasing their size not being practicable, other shafts were made under the specification that the elastic limit of the steel should be at least 105,000 pounds per square inch, its contraction of area 40 per cent, and its ball hardness uniform within 5 per cent. Shafts to meet such qualifications were made of chromium-vanadium steel containing 0.33 per cent C, 0.54 per cent Mn, 0.022 per cent P, 0.030 per cent S, 0.89 per cent Cr, and 0.24 per cent V. The ingot, which was 30 by 25 inches in section, was rolled to an 18 by 18 inch bloom or billet, and the shafts were forged therefrom. The shafts were heat-treated, and a test from one of them, about the average of all those made, pulled at Watertown Arsenal on a 2-inch by 0.505 diameter section, gave results as follows:

Results of tests of heat-treated chrome-vanadium steel shaft.

Elastic limit.	Tensile strength.	Elongation.	Contraction.	Ball hard- [illegible]
Pounds.	*Pounds.*	*Per cent.*	*Per cent.*	
105,260	127,310	15	46.2	278
....	....	....	....	283
....	....	....	....	278

These shafts met the specifications and proved satisfactory in service.

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RECOVERING ALLOYED ELEMENTS.

As the alloying elements of alloy steels are all more valuable than iron, some of them very much so, makers of alloy steels wish, of course, to save such elements in their alloy-steel scrap, if practicable without too great cost. The makers can save tungsten, copper, nickel, and cobalt, each of which has an affinity for oxygen and a heat of combustion with oxygen less than the similar properties of iron; however, the makers can not at present save the manganese, silicon, vanadium, titanium, calcium, magnesium, and chromium, each of which has an affinity for oxygen and a heat of combustion greater than the similar properties of iron. Plans have been proposed for saving some of them. Saving a part of these oxidizable elements in reworking the scrap is in some instances of no benefit, because in replacing the wasted part by the addition of a fresh quantity of the ferro-alloy saturated with carbon there will be too much carbon in the finished steel. This is particularly true of manganese in manganese-steel scrap. The carbon in the alloy-steel scrap is protected from oxidation by the alloying element while the latter is being oxidized. With low-carbon alloy available the case is different, as the content of carbon in the steel may then be controlled.

CERIUM PYROPHORIC ALLOY.

The pyrophoric metal used on cigar lighters and for igniters in miner's lamps might be considered as an alloy steel, as it consists substantially of 30 per cent of iron with 70 per cent of cerium. It was patented by Welsbach, whose name is identified with the Welsbach light. The striker is of the grade of hardened file steel with about 1.50 per cent carbon. The detached particles of the cerium-iron alloy take fire in the air, ignition being quickened no doubt by the heat generated in the impact of the striker.

CONCLUSION.

Further advance in the development of new alloy steels, as well as many new applications of those alloy steels already established, are to be expected. Trials are continually being made of new alloys of promise, some of which will doubtless win place in the list of useful alloy steels. Hadfield's iron alloy containing 5 per cent manganese and 15 per cent nickel, although not at present of use, may become so in the future, as its properties are rather remarkable.

As some of the alloys in steel, as well as any heat treatment it may have received, affect the carbon contained so that its effect in the color determination is changed, the regular practice in some steel-works laboratories is to make all carbon determinations gravimetrically by direct combustion of the whole sample with oxygen. This procedure avoids the uncertainties and errors of the color determination in analyzing heat-treated alloy steels.

ACKNOWLEDGMENT.

The author desires to make grateful acknowledgment of the cooperation of the following firms or individuals, who are some of the many who have contributed information for this report: Pond Machine Tool Co.; C. W. Leavitt & Co.; Bethlehem Steel Co.; Crucible Steel Company of America; Carbon Steel Co.; Firth-Stirling Steel Co.; Carnegie Steel Co.; Peerless Motor Car Co.; Society of Automobile Engineers; Fore River Engine & Ship Building Co.; Tabor Manufacturing Co.; United Steel Co.; Carpenter Steel Co.; Pennsylvania Steel Co.; Cambria Steel Co.; Columbia Tool Steel Co.; Edgar Allen Manganese Steel Co.; Primos Chemical Co.; Henry Disston's Sons Co.; Simonds Manufacturing Co.; Ford Mortor Co.; Perfection Spring Co.; Chrome Steel Works; Standard Roller Bearing Co.; Central Railroad of New Jersey; New York Central & Hudson River Railroad Co.; Delaware, Lackawanna & Western Railroad Co.; Pennsylvania Railroad Co.; R. R. Abbott; E. O'C. Acker; Westinghouse Electric & Manufacturing Co.; J. A. Mathews; F. V. Sargent; Hans Boker; W. A. Bostwick; B. E. Eldred; Dr. Paul R. Heyl; Illinois Steel Co.; W. M. Stein; and Leonard Waldo.

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DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

ABSTRACTS OF CURRENT DECISIONS
ON
MINES AND MINING

OCTOBER, 1914, TO APRIL, 1915

BY

J. W. THOMPSON

WASHINGTON
GOVERNMENT PRINTING OFFICE
1915

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First edition. July, 1913.

PREFACE.

This bulletin is the fourth of its kind to be published by the Bureau of Mines, the three preceding being Bulletin 61, Bulletin 79, and Bulletin 90.

The wide demand for the information contained in these bulletins has led the bureau to decide to issue similar bulletins with sufficient frequency to keep resaonably current the records of decisions of Federal and State courts of last resort on questions relating to the mineral industry.

The bureau will gladly welcome and consider any suggestions looking to improvement in the matter contained in these bulletins or the manner in which it is presented. The purpose of the bulletins will continue to be to improve directly or indirectly mining conditions and to promote the health and safety of miners by the prompt publication of decisions, and to this end it is desired that the bulletins reach all persons who are interested.

J. A. Holmes.

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The owner of minerals under a conveyance from the surface owner does not lose his right or his possession by mere nonusage of the minerals.

McBeth *v.* Wetnight (Indiana), 106 Northeastern, 407, p. 410, October, 1914.

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A complaint in an action to quiet title to minerals and to the mineral rights in certain land is insufficient in the absence of a direct allegation of possession, although it avers that the plaintiff from the time of receiving the conveyance assessed the mineral interest for taxes each year and paid the taxes thereon and exercised various acts of ownership of such mineral interests, in that he caused test pits to be dug on the land for the purpose of exploring the minerals and ascertaining their quality and extent, in going upon the land and looking after them from time to time, and that he accompanied or caused his agents to accompany prospective purchasers of such mineral interests and exhibited the same to such prospective purchasers in efforts to sell them.

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Big Sandy Coal Co. *v.* Ramey (Kentucky), 172 Southwestern, 508, p. 511, January, 1915.

SALE AND CONVEYANCE.

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Shrewsberry *v.* Pocahontas Coal & Coke Co., 219 Fed., 142, p. 147.

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Kentucky River Consolidated Coal Co. *v.* Frazier (Kentucky), 170 Southwestern, 986, p. 988, December, 1914.

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Carter *v.* Butler (Missouri), 174 Southwestern, 399 p. 404, March, 1915.

RESERVATION OF RENTS AND ROYALTIES—EFFECT AND EXTENT.

A conveyance by a landowner containing a reservation to the effect that if his wife survived him she was "to receive one-half of all rents from off the place, from all resources whatsoever," gives to the widow the right to receive rents and royalties from mines that were open and operated on the land at the time the conveyance was made; and the words "all resources whatsoever" necessarily include such mines, and the grantor intended that the widow should share equally not only in the income from the mines but in all other income and proceeds from the farm, regardless of the manner and form of payment, or the name by which it might be designated.

Saulsberry *v.* Saulsberry (Kentucky), 172 Southwestern, 932, p. 933, February, 1915.

SURFACE AND MINERALS—OWNERSHIP AND SEVERANCE.

SEPARATION OF MINERAL AND SURFACE ESTATES.

A conveyance by a landowner of the underlying coal with the privilege of mining and removing the same from under the land, effects a severance of the right to the surface from the right to such underlying coal; and when these rights are so severed by conveyance the presumption that the party having possession of the surface has possession of the subsurface also no longer prevails.

McBeth *v.* Wetnight (Indiana), 106 Northeastern, 407, p. 410, October, 1914.

RESERVATION OF MINERALS.

A deed conveying lands and excepting therefrom all coal or other minerals or mineral waters, sufficiently manifests an intent to except the minerals in the land from the operation of the deed and such a reservation of a proper subject matter of an exception is in law an exception though incapable of operation as a reservation, since it expresses an unequivocal intent not to part with the thing reserved.

Freudenberger Oil Co. *v.* Simmons (West Virginia), 83 Southeastern, 995, p. 988, December, 1914.

CONVEYANCE OF MINERALS—IMPLIED RIGHT TO MINE.

A grant of minerals under the surface of the land by the owner of the surface implies the right to mine such minerals by the sinking of shafts or boring of tunnels and the removal of minerals through such openings.

Himrod *v.* Fort Pitt Mining & Milling Co., 220 Fed., 80, p. 82, January, 1915.

GRANT OF MINERALS—RIGHT TO OPERATE ROADS UPON SURFACE.

A grant by a landowner of the underlying minerals implies the right to construct and operate roads and tram and railway tracks upon the surface for the use of the mine, to build air shafts, erect machinery, store water for the use of engine, and in general to do that which is reasonably necessary for the use of the thing granted; and it is not requisite to an implied grant that there be absolute physical necessity for the right demanded.

Himrod *v.* Fort Pitt Mining & Milling Co., 220 Fed., 80, p. 82, January, 1915.

SEVERANCE—EFFECT OF ADVERSE POSSESSION.

Where there has been a severance by deed of the surface of land from the underlying mineral the possession of the surface for more than 20 years does not carry with it the possession of such minerals

beneath the surface; and where such surface owner seeks to establish title to the underlying minerals by adverse possession as against one holding the legal title to such minerals he must prove possession of the minerals as such independent of his surface possession.

McBeth *v.* Wetnight (Indiana), 106 Northeastern, 417, p. 410, October, 1914.

TITLE TO MINERALS—ADVERSE POSSESSION.

Where there has been an actual valid severance by deed or by adverse possession of the title to the surface and the title to the minerals underneath, a mere cessation of the working or operation of a mine on the land by the owner thereof, or mere nonusage of the mineral, will not deprive the owner of the mineral of his right thereto or his possession thereof. To effect this there must be more than an abandonment or nonusage by the owner of the particular mine, and the proof must show an abandonment or disseizin of his possession of the mineral right, before the owner of the surface can ripen a title to such underlying minerals by adverse possession.

McBeth *v.* Wetnight (Indiana), 106 Northeastern, 407, p. 411, October, 1914.

TITLE—ADVERSE POSSESSION OF MINERALS.

It is essential in order to effect adverse possession of minerals, after severance in title from the surface, that the adverse claimant do some act or acts evincing a permanency of occupation and use, as distinguished from acts merely occasional, desultory, or temporary acts, that are suitable to the enjoyment and appropriation of the minerals so claimed, and hostile to the rights of the owner; but the mere possession of the surface after such severance does not give title to the minerals.

Birmingham Fuel Co. *v.* Boshell (Alabama), 67 Southern, 403, p. 404, December, 1914.

ADVERSE POSSESSION OF SURFACE—EFFECT OF MINERALS.

After severance of mineral in place from the surface, the possession of the surface is not possession of the minerals, as the effect of the severance is to create two closes adjoining but separate, and under such circumstances the acquisition of title to the surface by adverse possession of the surface does not result in the acquisition of title to the mineral interest in the land.

Birmingham Fuel Co. *v.* Boshell (Alabama), 67 Southern, 403, p. 404, December, 1914.

TITLE TO MINERALS NOT LOST BY NONUSAGE.

A person in the possession of the surface land is presumed to have possession of the subsoil also; but when by conveyance or reservation a separation has been made of the ownership of the surface from that

of the minerals below the surface, the owner of the former can acquire no title to the latter by his exclusive and continued enjoyment of the surface; nor does the owner of the minerals lose his right of possession by any length of nonusage; but to lose his right he must be disseized, and there can be no disseizin by an act which does not actually take the minerals out of his possession.

Shrewsbury *v.* Pocahontas Coal & Coke Co., 219 Fed., 142, p. 147.

POSSESSION OF SURFACE—EFFECT ON UNSEVERED MINERALS.

Possession under a general warranty deed of a tract of land described therein is the possession of all the elements of the land and of everything underneath the surface, unless the mineral products have therefore been separated from the ownership of the surface by a previous conveyance.

Big Sandy Coal Co. *v.* Ramey (Kentucky), 172 Southwestern, 508, p. 510, January, 1915.

MINERAL AND AGRICULTURAL LANDS.

The act of August 30, 1890 (26 Stat., 391), limited the amount of land to be acquired by any one person to 320 acres in the aggregate; and the subsequent act of March 3, 1891 (26 Stat., 1095, p. 1101), expressly limited the maximum amount of 320 acres that could be acquired by any one person to agricultural lands, and has no application to lands entered or sought to be entered under mineral lands laws.

Shenk *v.* Aumiller, 217 Fed., 969, p. 972.

GRANT OF TUNNEL RIGHT—IMPLIED RIGHT TO DEPOSIT TUNNEL WASTE ON SURFACE.

There are obvious degrees of necessity for the use of the surface in the conduct of subterranean operations, from the absolute necessity of sinking shafts or making other entrances to the mineral, to the practical necessities of business operations, such as the placing of steam engines and machinery at the mouth of the entrances, or constructing ponds of water to supply the engines, of laying and operating railways or tramways to bring in supplies and to carry out the ore, of storage of minerals on the surface pending sale, of assembling houses, stores, and shops for the use of miners, and such uses are implied in the grant if necessary to the operation of the property; and under this rule a grant of a right to bore or construct a tunnel may imply the right, as a reasonanle necessity, to use the surface for the deposit of waste and débris, brought from the tunnel or shaft; but such necessity is to be determined as a question of fact from the circumstances of each particular case.

Himrod *v.* Fort Pitt Mining & Milling Co., 220 Fed., 80, p. 83, January, 1915.

RIGHT TO DEPOSIT TUNNEL REFUSE ON SURFACE—QUESTION OF FACT.

Under a grant of a right to construct a tunnel for mining purposes, where the mouth of the tunnel was located at an elevation of some 10,000 feet above the sea level and upon a steep mountain side, and where it appeared that the owner of the surface, the grantor of the tunnel right, had long used the surface as a place of deposit for similar refuse brought from his own mine and from his portion of the same tunnel, the question of the reasonable necessity of the right of the tunnel grantee to deposit the tunnel refuse and débris on the surface is a question of fact and not of law.

Himrod *v.* Fort Pitt Mining & Milling Co., 220 Fed., 80, p. 84, January, 1915.

COAL AND COAL LANDS.

FRAUDULENT ACQUISITION UNDER HOMESTEAD LAWS.

The fact that lands are located in a well-known coal region and generally reputed to be coal lands, and where an extensive and expensive tunnel, slope, and other openings upon the lands had disclosed that they contained coal of such quality and quantity as to render them valuable for coal mining, and where a prospective purchaser from the patentee caused the lands to be examined by an engineer, who found and reported the tunnel and other openings disclosing the coal, is sufficient, as persuasive evidence at least, to charge such purchaser with notice or knowledge of the fraud of the patentee in obtaining coal lands under homestead law.

Washington Securities Co. *v.* United States, 234 U. S., 76, p. 78.

OPTION TO PURCHASE—CONSTRUCTION.

The owner of certain described coal lands gave to another an option to purchase such land, reserving to himself 5 acres of the surface and describing such 5 acres as the part thereof which had been in cultivation and extended back from the river front in such form as to include buildings, and stipulating that the boundaries of the 5-acre tract were to be definitely located within 60 days from the date of the instrument; but the failure of the owner to fix the boundaries of the 5-acre tract within the 60 days did not amount to a conveyance or an abandonment thereof to the optionee, as such an option is not like an exception from a deed which would be construed to convey the whole land regardless of the attempted reservation, though the condition was fully complied with as against the rights of the optionee, where the boundaries of the 5-acre tract were definitely fixed before the time the optionee was entitled to a deed under the option contract.

Rouse *v.* Riverton Coal & Development Co. (Oregon), 142 Pacific, 343, p. 344.

AGREEMENT RELATING TO COAL IN PLACE.

A contract regarding coal in place may be a sale absolute, a conditional sale, a lease in the ordinary acceptation of that term, or a mere license to mine or remove the mineral.

Gerard Trust Co. *v.* Delaware & Hudson Co. (Pennsylvania), 92 Atlantic, 129, p. 130, July, 1914.

CONTRACT FOR COAL LANDS—SPECIFIC PERFORMANCE.

Specific performance of an agreement by which an owner of the surface and other rights of coal lands agreed to deed the surface and such rights to a corporation to be formed and of which the grantor should be one of the corporators can not be defeated by proof of alleged representations as to the dividends the corporation would probably declare, the profits that would be derived from the business, as these are mere matters of opinion and the failure of such expectations to be realized in a speculative venture is no reason for refusing specific performance where the other party has furnished the capital and performed all the conditions of the contract on his part.

Bartley *v.* Big Branch Coal Co. (Kentucky), 169 Southwestern, 601, p. 602, October, 1914.

TITLE TO COAL—ADVERSE POSSESSION.

Subsequent to a severance by deed of the surface and underlying coal, the grantees under a deed sufficient to give color of title to the entire real estate, took actual possession thereunder of the real estate and of a mine being operated thereon, such possession while it continued extended to the coal under the entire tract conveyed as well as to the surface, and if continued for the required period would have developed into an absolute title to all the coal under the tract conveyed; but the subsequent abandonment of the mine broke the continuity of possession and thereby defeated the title to the coal by adverse possession as against the person holding the legal title of the coal under a former conveyance.

McBeth *v.* Wetnight (Indiana), 106 Northeastern, 407, p. 410, October, 1914.

SURFACE OWNERSHIP NOT ADVERSE TO MINERAL RIGHTS.

The possession of the surface owner of land is not adverse to the title of the owner of coal and minerals beneath the surface; but such surface possession may inure to the benefit of the owner of the underground mineral.

Shrewsbury *v.* Pocahontas Coal & Coke Co., 219 Fed., 142, p. 147.

SEVERANCE OF OWNERSHIP—RESERVATIONS IN DEED.

A deed describing land as containing 25 acres, more or less, the surface only, with the coal and mineral excepted, and with such exceptions as have heretofore been conveyed away, plainly imports a previous conveyance of the coal and minerals in the land and shows a complete severance of the surface and minerals.

Shrewsbury *v.* Pocahontas Coal & Coke Co., 219 Fed., 142, p. 146.

LIFE ESTATE IN SURFACE—EFFECT ON COAL.

A deed, by which the grantor purported to convey a designated tract describing the same by metes and bounds without making any reference to the coal thereunder, where the grantor at the time of the conveyance held under a deed conveying to her a life estate only in the surface, and there had been no determination as to the extent of her interest in the coal, conveys a life estate only and does not pass to the grantee any interest in the coal under the land described.

Dotson *v.* Norman (Kentucky), 169 Southwestern, 527, p. 530, September, 1914.

RESTRICTION AGAINST ALIENATION OF COAL.

Under a deed by which a father conveyed to his son certain coal lands in consideration that the grantee keep and maintain the grantor and his wife during their natural lives, and containing a provision that the second parties shall not sell or convey said land during the lifetime of the grantor without his consent, the grantees can not, during the lifetime of the grantor and without his consent, sell and convey the coal underlying the said tract of land, as such restriction applies as much to the underlying coal as to the surface itself.

Pond Creek Coal Co. *v.* Runyon (Kentucky), 170 Southwestern, 501, p. 503, November, 1914.
Kentucky River Consolidated Coal Co. *v.* Frazier (Kentucky), 170 Southwestern, 986, p. 987, December, 1914.

CONVEYANCE OF COAL IN VIOLATION OF RESTRICTIONS—TITLE.

A purchaser of coal underlying a certain tract of land is not an innocent purchaser for value where the deed to his grantor expressly provided that such grantor should not sell the underlying coal within the lifetime and without the consent of such original grantor, and the mere delay of the original grantor to assert the invalidity of the conveyance of the coal by his grantee will not defeat his right where the grantee of the coal received no title to the coal because the conveyance was in violation of the provisions of the original conveyance.

Pond Creek Coal Co. *v.* Runyon (Kentucky), 170 Southwestern, 501, p. 503, November, 1914.

MINERAL CHARACTER DETERMINED BY LAND DEPARTMENT.

An investigation and determination by the Land Department of the mineral character of land on an application for entry of land under the timber and stone act is conclusive, and a patent issued thereon will not be vacated or canceled on the ground of fraud on proof of the subsequent discovery of coal.

United States *v.* Primrose Coal Co., 216 Fed., 553, p. 557.

FRAUDULENT RESERVATION OF MINERALS—RIGHTS OF CREDITORS.

The fact that a corporation sold a large tract of land to an ignorant negro and fraudulently reserved to itself in the deed of conveyance the coal and oil underlying the land does not authorize or justify a creditor of the grantor, who subsequently recovered a judgment against the grantor corporation, levying upon and selling the estate in the coal and oil so fraudulently received, though there has been no reformation of the deed, where the creditor of the grantor had no lien upon the land and credit was not extended to him on the faith of the reservation in the deed and where it does not appear whether the debt was created before or after the execution of the deed.

Hyden *v.* Calames (Kentucky), 171 Southwestern, 186, p. 187, December, 1914.

POWER OF COURT TO SELL COAL ESTATE OF INFANT.

Under the statute of Kentucky a court may order the sale of any interest in land owned by an infant, whether legal, equitable, vested, or contingent, and under this statute a court may order the sale of coal under the land of an infant without selling the surface; but a court is not justified in ordering the sale of the mineral rights of an infant and giving the purchaser such rights in the surface as to destroy the value of the remaining estate, and where the purchaser would acquire such rights in the surface as would make it impossible for the owner to sell the surface except at a great sacrifice.

Hays *v.* Wicker (Kentucky), 171 Southwestern, 447, p. 448, December, 1914.

STONE LANDS.

EFFECT OF SUBSEQUENT DISCOVERY OF COAL.

The timber and stone act (20 Stat., 89) requires that when any valuable deposit of gold, silver, cinnabar, copper, or coal are known to exist on the land sought to be entered under such act, the entryman is only required to state such fact "as deponent verily believes," and a patent issued thereon will not be vacated on proof of subsequent discovery of coal where before the patent was issued an inves-

tigation was had as to the mineral character of the land and a special agent was sent to personally examine the land and after such examination reported that it was chiefly valuable for timber and stone.

United States *v.* Primrose Coal Co., 216 Fed., 553, p. 557.

OIL AND OIL LANDS—SALE AND CONVEYANCE.

WITHDRAWAL ORDER VALID AS AGAINST SUBSEQUENT OIL LOCATIONS.

The withdrawal order of the President of September 27, 1909, known as the "Temporary petroleum withdrawal No. 5," by which an area aggregating over 3,000,000 acres in California and Wyoming, containing petroleum deposits, was withdrawn from acquisition under the mining laws, is valid and binding as against subsequent oil locations.

United States *v.* Midwest Oil Co., 236 U. S., —, February 23, 1915.

TITLE—EFFECT OF WITHDRAWAL ORDER.

The withdrawal order of September 27, 1909, was ineffective as against a completed oil location and upon which the locator had expended a large sum of money and had discovered oil before the passage of the act of June 25, 1910 (36 Stat., 847).

United States *v.* McCutchen, 217 Fed., 650, p. 655.

OIL LOCATIONS—EFFECT OF SUBSEQUENT LOCATIONS.

Subsequent oil locations made for the purpose of protecting and strengthening the original location can not be regarded as fraudulent, though made in whole or in part by persons who had no intention of claiming the land or any interest in it for themselves.

United States *v.* McCutchen, 217 Fed., 650, p. 654.

POSSESSOR'S RIGHT TO OIL AND GAS.

Under the Louisiana Civil Code the right of a possessor in good faith to the benefit of the fruits of the land until it is claimed by its owner does not permit such possessor to extract the mineral, oil, and gas from the land and retain the proceeds; and the word "products" as used in the code is synonymous with fruits.

Elder *v.* Ellerbe (Louisiana), 135 Louisiana, —, 66 Southern, 337, November, 1914.

TERMINATION OF ESTATE.

A conveyance by deed of the oil and gas in and under a certain tract of land, in consideration of the payment of a certain stated sum, and upon condition for the payment to the grantor of another stated sum or reconveyance to him of the oil and gas within 90 days after

the completion of a well on the premises, or payment of another stated sum within two years from the date of the deed and payment of a stipulated purchase price or reconveyance after the completion of the well, but subsequent to the expiration of such two-year period, is conditional also upon the completion of such well within two years after the expiration of the two-year period expressly provided in the deed; and the estate granted ipso facto ceased on the expiration of four years without completion of a well, even though the last stated payment was made as contemplated.

Philadelphia Co., etc., *v.* Underwood (West Virginia), 83 Southeastern, 905, December, 1914.

PIPE LINE ACT—CONSTITUTIONALITY.

The Hepburn Act, regulating pipe lines, deals with commerce among the various States, and the fact that oils transported belong to the owner of the pipe line is not conclusive against the transportation being such commerce.

Pipe Line Cases, In re, 234 U. S., 548, p. 560.

PIPE LINE—OPERATION FOR PRIVATE USE.

When an oil company is simply drawing oil from its own wells across a State line to its own refinery for its own use and nothing more, it does not fall within the description of the Hepburn Act regulating pipe lines, the transportation in such case being merely an incident to its use at the end.

Pipe-Line Cases, In re, 234 U. S., 548, p. 561.

PIPE LINES—STATUTORY REGULATIONS.

The Hepburn Act was intended to reach the combination of pipe lines by which the Standard Oil Co. owned and controlled either all or a large part of the stock of the New York Transit Co., the National Transit Co., and the Ohio Oil Co., and the National Transit Co., which in turn owned nearly all the stock of the Prairie Oil & Gas Co.; but the provisions of the act are to apply to any persons engaged in the transportation of oil by means of pipe lines; and the words "who shall be considered and held to be common carriers within the meaning and purpose of this act" are not intended to cut down the generality of the previous declaration and mean that only those shall be held common carriers under the act who were common carriers in a technical sense; and while the act does not compel them to continue in operation, it does require them not to continue except as common carriers. The purpose of the act was to bring within its scope pipe lines that, although not technically common carriers, yet were cariers of oils offered, if only the offerers would sell at their price.

Pipe-Line Cases, In re, 234 U. S., 548, p. 559.

EMINENT DOMAIN.

APPROPRIATION OF LAND—TITLE TO MINERALS.

Eminent domain being an attribute of sovereignty unlimited by the Constitution, the State may, through its legislature, in the exercise of its high prerogative, authorize a public-service corporation to take any estate in land dictated by its sovereign will; and under the express terms of the statute of West Virginia a railroad company, by condemnation and compliance with all the provisions of the law, takes an estate in fee simple absolute in the land taken, including the oil and gas and other minerals in and under the same.

Hays *v.* Walnut Creek Oil Co. (West Virginia), 83 Southeastern, 900, p. 901, December, 1914.

PRIVATE PROPERTY—PRIVATE WAYS OF NECESSITY.

The constitution of Arizona provides that private property shall not be taken for private use except for private ways of necessity, or for drains or flumes on or across the lands of others for mining and other stated purposes; but the legislature has given no definition for the term "private ways of necessity" and the courts can not, in the absence of such a definition by the lawmaking body, define what shall constitute private ways of necessity, and accordingly a court can not say that a tunnel extended from one group of mining claims to another through an intervening mining claim privately owned, is a "private way of necessity," that may be appropriated for mining purposes.

Inspiration Consolidated Copper Co. *v.* New Keystone Copper Co. (Arizona), 144 Pac., 277, p. 279, November, 1914.

APPROPRIATION OF PRIVATE PROPERTY FOR MINING PURPOSES.

The constitution of Arizona authorizes the legislature to enact legislation providing for the condemnation of private property for private use, to wit, private ways of necessity, and for drains, flumes, or ditches, on or across the lands of others for mining purposes, and the private use for which property may be appropriated need not in any sense be a public use.

Inspiration Consolidated Copper Co. *v.* New Keystone Copper Co. (Arizona), 144 Pac., 277, p. 279, November, 1914.

APPROPRIATION OF RIGHT OF WAY—DAMAGES.

In appropriation proceedings by a railroad company for land for a right of way it is not proper in estimating damages to take into consideration a large tract of coal land not contiguous to the tract or tracts, parts of which were taken for the right of way.

Buchannon & Northern Railroad Co. *v.* Great Scott Coal & Coke Co. (West Virginia), 83 Southeastern, 1031, p. 1035, December, 1915.

DAMAGES FOR APPROPRIATION OF LAND—BURDEN OF PROOF.

In proceedings for the appropriation of the surface of coal lands for a right of way for a railroad, the owner of the land is entitled to the value of the land taken at the time of taking and to damages to the residue, deducting special but not general benefits, and in neither case has the owner the burden of showing general benefits; but the burden is upon the landowner to prove the value of the land taken and the damages to the residue, while the burden is upon the railroad company to prove special benefits in order to lessen the damages.

Buchannon & Northern Railroad Co. *v.* Great Scott Coal & Coke Co. (West Virginia), 83 Southeastern, 1031, p. 1038, December, 1915.

APPROPRIATION OF LAND FOR HIGHWAY—RIGHT TO SECOND WAY.

The statute of Iowa provides that any person or corporation owning or leasing any lands not having a public or private way thereto may have a public way to any railway station, street, or highway established over the land of another; and provides also that any owner, lessee, or possessor of lands having coal or other minerals thereon, who has paid the damages assessed for roads established as provided, may construct, use and maintain a railway thereon for the purpose of reaching and operating any mine on such land and of transporting the products thereof to market; and this gives the person or corporation named his choice as between a highway and the railway connection and no limitation is put upon his use of such way as he may acquire and he may use it as a wagon way or railway, or both; but having acquired the one or the other he can not condemn for outlet purposes; and accordingly a coal company having a private way from its coal mine and lands to a public highway, is not entitled to condemn a right of way for a railroad switch in addition to its private way.

Fisher *v.* Maple Rock Coal Co. (Iowa), 151 Northwestern, 823, p. 824, March, 1915.

MINING TERMS.

APPLIANCES.

Motor trucks in the main haulageway of a coal mine constitute an appliance for the removal of coal.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 942, April, 1915.

APPLIANCES OF TRANSPORTATION.

Appliances of transportation as applied to a coal mine include the motor tracks, roadbed, cars, and motors used for the removal of coal from the mine.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 942, April, 1915.

BLOCKHOLER.

A blockholer in a mine is a person whose duty it is to break up and reduce to safe and convenient size, by blasting or otherwise, any large blocks or pieces of rock that have been blown down by the miners; and in the prosecution of the work his duty is to go from one level of a mine to another in the territory assigned to him, following up the work of the miners, looking for and breaking up such pieces, and in connection with that work starting them, as well as other large pieces, which can be moved without breaking, down the incline to where the trammers can conveniently handle them, and at times when these duties are performed he also passes down dirt and works the material toward the place of loading the cars to facilitate the work of the trammers.

Mesich *v.* Tamarack Mining Co. (Michigan), 151 Northwestern, 564, p. 563, March, 1915.

CAP.

A "cap" is a square piece of plank or block wedged between the top of posts and the roof of a mine the better to hold the roof.

Big Branch Coal Co. *v.* Wrenchie (Kentucky), 170 Southwestern, 14, p. 16, November, 1914.

CUT.

The word "cut" may have a meaning other than that employed in mining, but when it is used in conjunction with "shaft" and "drift" it means a surface opening in the ground intersecting a vein, and is never intended to apply to a ditch or trench temporarily open for the purpose of laying sewer pipe.

McLaughlin *v.* Bardsen (Montana), 145 Pacific, 956, January, 1915.

"DEAD MAN."

A "dead man" in a mine is a wooden block used to guard the mouth of a mine against runaway cars.

Connors Weyman Steel Co. *v.* Kilgore (Alabama), 66 Southern, 609, p. 612, November, 1914.

GASOLINE.

Gasoline is a colorless, inflammable fluid, the first and highest distillant of crude petroleum, representing the lightest portions of crude oil, is extracted from it by distillation; and being the most volatile component of petroleum, it readily separates from it and in the process of distillation is the oil drawn off at the lowest temperature.

Locke *v.* Russell (West Virginia), 84 Southeastern, 948, p. 950, April, 1915.

GAS WELL.

The words "gas well" used in an oil and gas lease mean a well having such a pressure and volume of gas, taking into account its proximity to market, as could be operated profitably and the gas utilized or disposed of commercially.

Prichard *v.* Freeland Oil Co. (West Virginia), 84 Southeastern, 945, p. 946, April, 1915.

HEADER.

Headers are pieces of plank—longer than a cap—extending over more of the roof and supported by two props, one at each end.

Big Branch Coal Co. *v.* Wrenchie (Kentucky), 170 Southwestern, 14, p. 16, November, 1914.

MINE—COAL MINE

The terms "mine" and "coal mine" are intended to signify any and all parts of the property of a mining plant, either on the surface or underground, that contribute directly or indirectly under one management to the mining or handling of coal.

Hakanson *v.* La Salle County Carbon Coal Co. (Illinois), 106 Northeastern, 617, p. 618, October, 1914.

PILLARS OR STUMPS.

Pillars or stumps in a coal mine are the natural supports left in the mine for the purpose of supporting the roof.

Northeast Coal Co. *v.* Hunley (Kentucky), 174 Southwestern, 732, March, 1915.

PROP.

By "prop" is meant an upright post surmounted by a square block wedged between the top of the post and the roof.

Big Branch Coal Co. *v.* Wrenchie (Kentucky), 170 Southwestern, 14, p. 16, November, 1914.

Eagle Coal Co. *v.* Patrick (Kentucky), 170, Southwestern, 960, p. 961, December, 1914.

PUSHER—JIGGER BOSS.

"Pusher" or "jigger boss" is a term used in mining parlance to designate one who is engaged for the purpose of encouraging or hastening the miners.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 908, December, 1914.

SHAFT.

The term "shaft" as used in the Illinois mines and miners act means any vertical opening through the strata which is or may be used for the purpose of ventilation, escapement, or for the hoisting or lowering of men and material in connection with mining coal.

Hakanson *v.* La Salle County Carbon Coal Co. (Illinois), 106 Northeastern, 617, p. 618, October, 1914.

STOPE.

A stope in a mine is an excavation made in removing the ore which has been opened up or made accessible by shafts and drifts or levels.

Mesich *v.* Tamarack Mining Co. (Michigan), 151 Northwestern, 564, p. 565, March, 1915.

STRIKE.

Shutting down a mine by calling out men in obedience to their obligation as members of the United Mine Workers of America is what is known as a "strike."

Mitchell *v.* Hitchman Coal & Coke Co., 214 Fed., 685, p. 705.

TIMBERING.

By "timbering" is meant the protecting against falls of roof formation of a mine, by means of horizontal timbers or caps extending across the passageway just under the roof, the ends of such timbers resting upon vertical timbers or posts.

Eagle Coal Co. *v.* Patrick (Kentucky), 170 Southwestern, 960, p. 961, December, 1914.

TRAMMER.

Trammers in a mine are common laborers, who, working chiefly with shovels, load the ore and dirt or rock into tram cars and push them out from the stope to where they are attached to cars which haul them to the shaft; and in loading the cars they also at times, when conditions require, go up or back into the stope and pass dirt toward the cars.

Mesich *v.* Tamarack Mining Co. (Michigan), 151 Northwestern, 564, p. 565, March, 1915.
See Zap *v.* Newport Mining Co. (Michigan), 151 Northwestern, 554, March, 1915.
Sabela *v.* Newport Mining Co. (Michigan), 151 Northwestern, 598, March, 1915.

MINING CORPORATIONS.

CORPORATE POWERS—SURRENDER OF COAL LEASE.

An express surrender of a coal lease by a mining corporation requires corporate action; and the president of such a corporation has no inherent power to surrender a mining lease belonging to the corporation.

Laing *v.* Price (West Virginia), 83 Southeastern, 497, p. 499, October, 1914.

POWER OF DIRECTORS TO DISPOSE OF PROPERTY.

The general incorporation act of Delaware (22 Delaware Laws, 394), authorizing a mining corporation to buy, sell, and deal in mines, acquired by exchange of property, and sell or otherwise dispose of and

deal in shares of other corporations, and the articles of incorporation of a mining company providing that the directors shall have the power to sell, assign, and transfer and convey and otherwise dispose of a part of the property, assets and effects of the corporation, less than the whole or substantially the whole thereof, on such terms and conditions as they may deem right and just, without assent of the stockholders, clothes the directors of a mining corporation with the power, when acting in good faith and with no personal advantage to themselves to exchange the mining property of the corporation for stock in another corporation, and the judgment of the directors as to the wisdom of the exchange will not be reviewed, where there is no evidence of inadequacy of consideration for the sale.

Butler *v.* New Keystone Copper Co. (Delaware), 93 Atlantic, 380, p. 382, February, 1915.

KNOWLEDGE OF DIRECTORS—ACQUIESCENCE NOT A RATIFICATION.

Where a secretary and manager of an oil company attempted to cancel an existing contract and made a settlement with a well driller and thereupon notified a majority of the directors of the corporation of the cancellation and settlement, the mere silence and acquiescence of the individual directors will not amount to a ratification of the act of the secretary and manager, as such a ratification by the directors could only be accomplished by a formal act on the part of the members when in session, and in such case the mere silence of the individual directors can not operate as an estoppel.

Blair *v.* Brownstone Oil Refining Co. (California), 143 Pacific, 1022, p. 1023.

POWERS OF MAJORITY OF STOCKHOLDERS—RIGHTS OF MINORITY.

Under the statute of Montana the holders of two-thirds of the issued stock of any Montana mining corporation may by vote authorize a sale of all the corporate property to the same extent as if all stockholders had consented thereto, and upon any such sale the mining corporation is dissolved and wound up as in other cases. A minority stockholder is powerless to prevent such a sale, as every stockholder consents to this statutory provision when he becomes a stockholder and it is part of his contract with the other stockholders and the corporation.

If a sale by two-thirds of the stockholders is fraudulent a minority stockholder may avoid the sale on the ground of fraud, or he may have an appraisal of his stock and compel a payment of its appraised value; but if he elects to have an appraisal of his stock he can not thereafter resort to his remedy for the fraud.

Wall *v.* Anaconda Copper Mining Co., 216 Fed., 242, p. 243.

POWER OF DIRECTORS TO DISPOSE OF ALL THE PROPERTY.

Neither the directors nor the stockholders of a prosperous mining corporation have power to sell all, or substantially all, the property of the corporation over the objection of the holder of a single share of stock; but if the business is unprofitable, the enterprise hopeless, and the mine a disappointment, and the further development unprofitable and unwise, then the holders of a majority of stock may, even against the dissent of the minority, sell all the property of the corporation with a view of winding up the corporate affairs. And where the charter provisions are broad enough to include such sale within the corporate purpose, it will be within the power of the majority of the stockholders to so sell, in any event, as against the protest of a minority of the stockholders.

Butler *v.* New Keystone Copper Co. (Delaware), 93 Atlantic, 380, p. 382, February, 1915.

CAPITAL STOCK NOT PAYABLE IN DEPRECIATED PROPERTY.

The laws of Missouri do not contemplate or permit a corporation to be formed with a fictitious capital stock merely by designating in the articles of incorporation that the capital stock consists of property of a named valuation; and stockholders of a mining corporation can not escape personal liability on their stock subscription by showing that the articles of incorporation state that the capital stock was fully paid in certain mining property taken at a stated value, and a judgment creditor of the corporation seeking to enforce the personal liability of stockholders is not estopped to claim that the stock is not fully paid up.

Rogers *v.* Stag Mining Co. (Missouri Appeals), 171 Southwestern, 676, p. 679, December, 1914.

RIGHT OF STOCKHOLDER TO EXAMINE PROPERTY.

A stockholder in a mining corporation is entitled to examine the mining property of the corporation and this includes the right to be accompanied by an expert.

Hobbs *v.* Davis (California), 143 Pacific, 733, p. 734.

SALE OF STOCK—SPECIFIC PERFORMANCE.

In an action by the purchaser of mining stock to compel the specific performance of a contract of sale by another stockholder and the transfer to him of the shares of stock under the agreement, a court of equity will not decree the specific performance of the agreement where the purchase price was $255,000 for stock of the intrinsic value of $624,000, and where the purchaser had paid only $1,000, and he had failed to make other payments according to the terms, and where time was specified to be of the essence of the contract, and

the sums paid on the contract were to be forfeited in case of the failure of the purchaser to complete the agreement; and the buyer can not justify his default in such payments on the ground that the seller had refused him the privilege of examining the mining property, such refusal being the alleged efficient cause of the buyer's default, where the buyer himself was a stockholder and had the right as such to inspect the mining property, and where the inspection of the property was not made a condition to the performance of the buyer's agreement.

Hobbs v. Davis (California), 143 Pacific, 733, p. 734.

FRAUDULENT SALE OF MINING STOCK—DUMP PART OF MINE—PRESUMPTION AS TO VALUE.

In an action for damages for false and fraudulent representations in the sale of stock of a mining corporation, and where the false statement was to the effect that there was a half million dollars' worth of ore on the dump ready to mill and sufficient to run a mill from three to five years, on proof that the officers of the corporation reported the mine as worthless, together with proof tending to show an abandonment of the property, a jury may assume that the dump, which was a part of the mine, was also worthless.

Rogers v. Rosenfeld (Wisconsin), 149 Northwestern, 33, p. 36, October, 1914.

FRAUDULENT SALE OF MINING STOCK—DAMAGES.

In an action for damages for false and fraudulent representations in making a sale of stock in a mining company, where the stock was falsely represented to be worth 50 cents on the dollar, and where the proof showed that the market value of such stock was nothing, or not to exceed 5 cents per share, the defendant can not complain where the jury found damages in favor of the plaintiff of less than 50 cents per share.

Rogers v. Rosenfeld (Wisconsin), 149 Northwestern, 33, p. 36, October, 1914.

SALE OF MINING STOCK—LIABILITY FOR FRAUDULENT REPRESENTATIONS.

The owner of stock in a mining corporation who, for the purpose of inducing another to purchase the stock, stated to him that he had just returned from the mining property and that there was ore enough on the dump at the mine to run a mill from three to five years, that it was high-grade ore, and that there was a half million dollars' worth of ore on the dump ready to mill, was held liable on proof that the representations were false, and such representations, when relied upon, do not constitute a mere expression of opinion, but were statements of facts on which the purchaser had a right to rely.

Rogers v. Rosenfeld (Wisconsin), 149 Northwestern, 33, p. 35, October, 1914.

SALE OF MINING STOCK—FRAUD—PERSONAL INVESTIGATION BY PURCHASER.

In an action on a promissory note given in payment of the purchase price of mining stock, the maker of the note can not defend on the ground of fraud and false representation where the evidence shows that the purchaser of the stock, the maker of the note, made the purchase on the result of his own investigation and not in reliance on the statements made by the payee or in the company's prospectus; and the statute of Connecticut imposing a penalty for making false statements as to the value of stocks, bonds, or property has no application where the evidence shows that the purchaser did not rely upon and was not misled or deceived by the statement of the seller.

Meech *v.* Malcolm (Connecticut), 92 Atlantic, 657, p. 659, December, 1914.

PURCHASE OF STOCK—PURCHASE SUBJECT TO PRIOR AGREEMENT TO SELL.

A purchaser of the stock and bonds from a stockholder of a mining corporation with actual notice that his vendor, the stockholder, has by contract bound himself to transfer such stock and bonds to another and that suit was then pending for the specific enforcement of such contract, takes title to such stock and bonds subject to the terms of the vendor's original contract and the judgment rendered in the pending suit.

Peoples Bank, etc., *v.* Columbia Collieries Co. (West Virginia), 84 Southeastern, 914, p. 916, April, 1915.

CONTRACT BY SOLE STOCKHOLDER FOR SALE OF PROPERTY—SPECIFIC PERFORMANCE.

A person who owns all the stock in a mining corporation and who has contracted with another, on a sufficient consideration, to cause the real estate of the corporation to be conveyed to such other contracting party may be compelled in an action for specific performance of the contract, no rights of creditors of the corporation being involved, to complete the contract and cause the property to be conveyed according to its terms.

Peoples Bank, etc., *v.* Columbia Collieries Co. (West Virginia), 84 Southeastern, 914, p. 917, April, 1915.

INVALID ISSUE OF STOCK—RIGHT TO VOTE.

The transferee of stock of a mining corporation, who takes the same with notice and knowledge that the stock was illegally issued by the corporation to itself, may be prevented from voting such stock at a corporate election.

Deal *v.* Erie Coal & Coke Co. (Pennsylvania), 93 Atlantic, 829, p. 832, January, 1915.

JUDGMENT AGAINST MINING CORPORATION—CONTINUANCE OF LIEN.

A judgment against an insolvent corporation taken for materials and supplies furnished the bondholders while in possession and used for the purpose of preserving the property as security for the bonds, will continue to be a lien upon the property of the corporation where the bondholders purchased the property under foreclosure proceedings and thereupon organized a new company as a holding company to take the title to the property, but no consideration was paid for the purchase of the property by the new company except the interest its incorporators owned as bondholders.

Spadra-Clarksville Coal Co. *v.* Kansas Zinc Co. (Kansas), 145 Pacific, 571, January, 1915.

RAILROAD COMPANY LIABLE FOR BONDS OF MINING COMPANY.

A railroad company owning the controlling interest of an insolvent coal company entered into a plan of reorganization of the coal company by which the new company issued bonds for the purpose of raising funds to carry on its enterprise, and at the same time executed a lease of its mining property. The railroad company thereupon entered into a contract with the lessee by which it agreed to pay a stated sum per ton for a stated number of tons per year for coal shipped by it, the payments to be applied in discharge of the coal company's bonds.

The coal lessee complied with its part of the agreement, was ready to, and did mine the minimum amount of coal. The railroad company failed to furnish cars sufficient to transport the coal mined, and it was held liable in damages in an action by the trustee of the bondholders for losses sustained by them for its failure to perform its part of the agreement and furnish cars necessary to transport the coal mined; and it was no defense for the railroad company, against its own wrongdoing, to claim that under the requirements of the Interstate Commerce Commission and the State railroad commission it was required to distribute its cars among all coal operators on its line in the ratio of their relative needs, as the railroad owned and controlled the coal company and was manipulated by the railroad company for its own purposes and in its own interests.

Wheeling & Lake Erie Railway Co. *v.* Carpenter, 218 Fed., 273, p. 280.

PURCHASE OF ASSETS OF ONE BY ANOTHER—LIABILITY.

Where one corporation transfers all of its assets to another and practically ceases to exist, without having paid its debts, the purchasing corporation takes the property subject to an equitable lien or charge in favor of the creditors of the selling corporation; and the same rule applies where one corporation goes out of existence by being

merged into or absorbed by another, and in all such cases the liabilities of the old corporation are enforceable against the new, on the principle that equity impresses a lien on the property thus taken over for the benefit of the creditors of the old concern. But the rule does not apply where one corporation merely sells a part of its property to another and does not cease to exist, and in such case there would be no liability on the part of the purchasing corporation for debts of the seller.

Carter Coal Co. *v.* Clouse (Kentucky), 173 Southwestern, 794, p. 796, March, 1915.

PURCHASING CORPORATION NOT LIABLE FOR DEBTS OF SELLER.

The fact that one mining corporation purchased some 15,000 acres of coal lands of another corporation and operated a mine on the land purchased is not sufficient to make the purchasing corporation liable to an injured miner for injuries received prior to the time of the purchase, where both the selling and buying corporation continued their existence as separate corporations and were not merged into one corporation; and the fact that one person is a stockholder in both corporations is of no weight in determining the liability of the purchasing corporation.

Carter Coal Co. *v.* Clouse (Kentucky), 173 Southwestern, 794, p. 794, March, 1915.

FAILURE TO PAY CORPORATION TAX—LIABILITY AND FORFEITURE OF FRANCHISE.

The statute of Colorado imposes an annual corporation tax on corporations of that State and provides that nonpayment of the taxes shall work a forfeiture of the corporate franchise, and provides also that an action of debt will lie for the recovery of the tax. In an action of debt by the State against a mining corporation for taxes overdue the corporation can not defend on the ground that its failure to pay the taxes for the first year operated as a forfeiture of its franchise, where it appears that the corporation continued in business for the succeeding years and was still exercising its right and still conducting its business.

Pinnacle Gold Mining Co. *v.* People (Colorado), 143 Pacific, 837, p. 839.

BREACH OF CONTRACT TO FINANCE.

Where shares of stock of a mining corporation were issued and delivered to a person in consideration that he finance the corporation and advance money for the working of its mines, it is no defense in an action against him for a breach of the agreement to show that he had contracted with and paid a third person to furnish the money and that such third person had failed to do so.

Cerro Cobre Development Co. *v.* Duvall (Arizona), 147 Pacific, 695, p. 699, March, 1915.

LIABILITY FOR MALICIOUS PROSECUTION.

A coal mining corporation may be liable in an action for damages for malicious prosecution though such prosecution was procured to be made by its superintendent acting in furtherance of the business of the corporation; and the person or corporation starting the malicious prosecution is liable for its continuance and for the malfeasance of the officer making the arrest.

Lyons *v.* Davy Pocahontas Coal Co. (West Virginia), 84 Southeastern, 744, March, 1915.

LIABILITY FOR TORT—PARTICULAR ACTS AS EVIDENCE.

The fact that a mining corporation paid the compensation of certain deputy constables and peace officers appointed by the court of quarter sessions of the county, and which were appointed to perform the duties of police officers, and where it appears that the services of such deputies were not performed on the property of the mining corporation but on the public highway, is not sufficient to render the mining corporation liable for damages for assaults and batteries committed by such deputies and police officers.

Ruffner *v.* Jamison Coal & Coke Co. (Pennsylvania), 92 Atlantic, 1075, January, 1915.

INSOLVENCY—ACT OF BANKRUPTCY.

Where a mining corporation had executed a chattel deed of trust covering its entire property, and because of insolvency had failed to make payments as provided in the chattel deed of trust and the trustee had taken possession of the property and had advertised the same for sale, the failure of the mining corporation or of the trustee to vacate a judgment and the levy of the execution within five days prior to the sale did not constitute an act of bankruptcy and did not give the execution creditor a preference.

Moark-Nemo Consolidated Mining Co., In re, 219 Fed., 340.

BANKRUPTCY—RECEIVER—PREFERENCE CLAIM FOR SERVICES.

A receiver of a mining corporation who was a capable mining man and superintendent of the company's business and was engaged at the mine for two months before the company went into the hands of the receiver is entitled to preference for the reasonable amount of his services.

German National Bank *v.* Young (Arkansas), 169 Southwestern, 1178, p. 1180, July, 1914.

INSOLVENCY—PERSONAL LIABILITY OF STOCKHOLDERS.

The obligation arising on an implied contract of a corporation to pay for coal wrongfully mined from the property of another and not paid for is a "debt unpaid" within the meaning of section 1204 of

the General Statutes of Kansas, 1889, providing that if any corporation be dissolved leaving debts unpaid suit may be brought against any person or persons who were stockholders at the time of such dissolution; but any such claim is barred by the three years' statute of limitations, and a suit against a stockholder for such a claim can not be maintained when four years have elapsed since the corporation suspended business.

Abernathy *v.* Loftus (Kansas), 147 Pacific, 818, p. 820, April, 1915.

MINING CLAIMS.

GENERAL FEATURES.

MINING LOCATIONS ON APEX OF VEIN.

The course of a vein appearing on the surface is the course of its apex, and as a miner is required to locate his claim along the vein this means that he must locate it along the outcrop or course of the apex if it is found impracticable for him to locate it along the strike of the vein.

Stewart Mining Co. *v.* Bourne, 218 Fed., 327, p. 329.

OWNERSHIP OF ORE—PRESUMPTION.

Presumptively all ore bodies found beneath the surface of an abandoned mining claim belong to the owner of the claim.

Stewart Mining Co. *v.* Bourne, 218 Fed., 327, p. 328.

CONTRACT TO CONVEY MINING CLAIMS—EQUITABLE RIGHTS AND TITLES.

The owner of certain mining locations and holding options to purchase other mines entered into a contract with a corporation by which, in consideration of the assignment of the capital stock of the corporation, he promised and agreed to transfer such mining locations and options to the corporation; and when pursuant to such contract the corporation did assign to him its shares of the capital stock, the property in the mining locations and in the options will be deemed to have passed to the corporation under the rule that equity will regard that as actually done which ought to have been done; and the person so receiving the shares of stock can not thereafter transfer a valid title to the mining locations and to the options to another corporation with notice and knowledge of the agreement and of the fact that the stock had been transferred; and the contracting corporation may compel a conveyance of the mining locations and the transfer of the options by the corporation or person so receiving the same.

Cerro Cobre Development Co. *v.* Duvall (Arizona), 147 Pacific, 695, p. 699, March, 1915.

RIGHTS OF HEIRS—POWER OF ADMINISTRATOR TO CONVEY.

The equitable title to mining claims held under a contract of purchase, passes, on the death of the purchaser, to his heirs at law subject to be divested by the administrator of the estate for the purpose of paying debts and expenses of administration; but if there is ample personal property in existence to pay such debts and expenses without resort to the real estate, then the administrator has no authority to dispose of the property in such mining claims; and any attempt to forfeit or waive the title of the heirs at law to such equitable right in such mining claims would be unavailing, and the unauthorized deed of the administrator would not have the effect of divesting the title of the heirs at law in the mining property.

Costello *v.* Cunningham (Arizona), 147 Pacific, 701, p. 708, March, 1915.

AGREEMENT TO PURCHASE—TRUST—TENANCY IN COMMON.

An agreement between two persons by which they were to purchase mining claims and acquire title to others by location, title to all of which was to be taken in the name of one of the contracting parties in trust for the use and benefit of both, makes the one receiving the title trustee for both, and also creates a relation of tenancy in common of the property.

Costello *v.* Cunningham (Arizona), 147 Pacific, 701, p. 708, March, 1915.

LOCATION NOTICE AND CERTIFICATE.

RECORD—RECITALS AS EVIDENCE.

The record of a location notice, reciting the citizenship of the locators, the fact of discovery of mineral, and that the location had been marked upon the ground, is not even prima facie evidence of the truth of the recital, for the reason that no such facts are required to be stated in any of the statutory notices, but such facts in a controversy over the location must be established by proof outside of the location certificate.

Childers *v.* Laham (New Mexico), 142 Pacific, 924, p. 925.

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Under the Federal statute and the statute of New Mexico the record of a mining location is required to contain nothing more than the names of the locators, the date of location, and the description of the claim by reference to some natural object or permanent monument as will identify the claim; but the posting of a location notice, the discovery of mineral, and marking on the ground of the boundaries of the claim are not required to be recited in the location notice.

Childers *v.* Laham (New Mexico), 142 Pacific, 924, p. 925.

ASSESSMENT WORK—CONSIDERATION.

PROOF OF ANNUAL LABOR.

The opinion of a witness, not qualified as an expert as to the value of annual labor performed on a mining claim, can not be prejudicial where the court instructs the jury as to the legal rights of the parties as to the amount of labor necessary to be done in order to hold the claim.

Cache Creek Mining Co. *v.* Brahenberg, 217 Fed., 240, p. 242.

ASSESSMENT WORK PERFORMED BY STRANGER.

Work on a mining claim performed by a brother of the deceased owner of such claim, for the purpose of keeping the claim alive, and making assets for the estate of the deceased brother, out of which a claim held by him against the estate could be paid, will not save the claim from relocation, where the executors, legatees, and devisees under the will of the deceased owner refuse to recognize and adopt the work done as assessment work upon the claim, and where, after the expiration of the year and subsequent to the date of relocation, they began the performance of independent assessment work.

McDonald *v.* McDonald (Arizona), 144 Pacific, 950, p. 955, December, 1914.

COOWNER PREVENTED FROM PERFORMING.

The interest of a coowner in a mining claim can not be forfeited, as provided for by section 2324 of the Revised Statutes of the United States and section 1426–o of the Civil Code of California, by another coowner, where such coowner or the persons or corporations for whom he held an interest in the mining claim forcibly prevented him from completing the assessment work and forcibly ejected and drove him from the mining claim while in the act of performing the assessment work.

Thompson *v.* Pack, 219 Fed., 624, p. 625, December, 1914.

RELOCATION.

FAILURE TO PERFORM ASSESSMENT WORK—ENTRY NOT A TRESPASS.

A brother of a deceased owner of a mining claim who, before the expiration of the year, went upon the claim for the purpose of performing the assessment work and to prevent forfeiture and relocation, and who during the time and after the expiration of the year occupied a house on the claim, can not be regarded as a trespasser where the executors and legatees and devisees under the will of his deceased brother refused to recognize or adopt the work done by him as assessment work and where after the expiration of the year

and before other work was commenced, he relocated the mining claim for his own benefit and where it appeared that he occupied the house with the consent of such executors, legatees, and devisees.

McDonald *v.* McDonald (Arizona), 144 Pacific, 950, p. 955, December, 1914.

PERSON ENTERING TO RELOCATE NOT A TRESPASSER.

An unpatented mining claim under section 2324 of the Revised Statutes of the United States is subject to relocation if the required amount of assessment work has not been performed; and a person who enters upon the claim after midnight of December 31 for the purpose of making a relocation because of the failure to perform the assessment work for the preceding year is not a trespasser.

McDonald *v.* McDonald (Arizona), 144 Pacific, 950, p. 956, December, 1914.

FORFEITURE.

PLEADING FORFEITURE.

In a suit to determine an adverse claim to a mining location it is sufficient in pleading a forfeiture of the rights of the plaintiff to aver "all of plaintiff's right to and in said claim became forfeited and the said claim and all of it became a part of the public domain, subject to location according to law as mineral land," and especially in connection with the further averment that the plaintiff had not performed the annual labor required by law for a period of three years or more.

Cache Creek Mining Co. *v.* Brahenberg, 217 Fed., 240, p. 241.

FORFEITURE OF COOWNER'S INTEREST.

Section 2324 of the Revised Statutes of the United States and section 1426–o of the civil code of California provide for the forfeiture of a part interest in a mining claim on the failure of the coowner to perform or to contribute to the performance of the assessment work.

Thompson *v.* Pack, 219 Fed., 624, p. 625, December, 1914.

EXTRALATERAL RIGHTS.

VEIN TERMINATING WITHIN LOCATION.

A mining locator can not enforce extralateral rights on a vein that enters his claim through one of its side lines and in its course or strike does not pass out of the claim at all.

Stewart Mining Co. *v.* Bourne, 218 Fed., 327, p. 329.

POSSESSORY RIGHTS.

ADVERSE POSSESSION—PROOF OF LOCATION.

An adverse possession of a mining claim for the statutory period is not available to a plaintiff in an action of ejectment unless the location is completed and proved; and an admission in the answer of a defendant in such an action of the existence of a valid location by the plaintiff's grantors is not available to the plaintiff if he fails to call it to the attention of the trial court, otherwise such an admission might be sufficient to avoid the necessity of proof of location.

Childers *v.* Laham (New Mexico), 142 Pacific, 924, p. 925.

INJUNCTION TO PREVENT CLOUD ON TITLE.

A coowner of a mining claim who while in the act of performing assessment work was forcibly prevented from completing the same and was forcibly ejected and driven from the claim by his coowner, or by persons or corporations for whom the latter held an interest in the claims, may enjoin such coowner from placing on record the notice of forfeiture with the affidavit of service thereof, as provided for by section 1426–o of the Civil Code of California, as this would constitute a cloud on owner's interest.

Thompson *v.* Pack, 219 Fed., 624, p. 626, December, 1914.

ADVERSE POSSESSION—PAYMENT OF TAXES.

Section 2405, Revised Statutes of Utah (1908), provides that all mines and mining claims shall be taxed at the price paid the United States therefore, unless the surface ground or some part thereof is used for some purpose other than mining and has a separate and independent value, in which case it shall be taxed at its value for such other purpose. Section 2868, Compiled Laws of Utah (1907) provides that adverse possession of real estate can not be established unless it has been occupied and claimed for a period of seven years continuously and the occupant or his grantors has paid all taxes levied and assessed upon such land. But this latter section does not prevent the occupant of a mining claim from obtaining title by adverse possession where the surface of such mining claim was valuable for purposes other than mining and had been assessed for taxes because of its value for such other purposes, and persons other than the occupant had paid such taxes, and where the descriptions of the parts of the surface so assessed for other purposes were so defective as to make the assessment invalid.

Utah Copper Co. *v.* Chandler (Utah), 142 Pac., 1119.

OWNERSHIP OF VEIN OR LODE—CONCLUSIVENESS OF JUDGMENT.

In an action to determine the ownership and possession of a certain vein or lode of ore, a judgment awarding the property to the plaintiff is not conclusive in a subsequent suit by the same plaintiff against a lessee of the defendant in the original action, where such lessee took possession of the property long prior to the institution of the original suit.

Doctor Jack Pot Mining Co. *v.* Marsh, 216 Fed., 261.

SALE AND TRANSFER.

FRAUD IN PURCHASING—RECOVERY.

An action in the nature of indebitatus assumpsit or debt may be maintained for the difference between $25,000 and $625 against a person who, pretending and professing to act for himself, and others, falsely and fraudulently represented to them that he could purchase certain mining claims for the sum of $25,000 and relying upon such representations the persons to whom the representations were made advanced the sum of $25,000 for the purchase of the mining claims, where the party making the purchase paid but $625 for the mining claims and appropriated the remainder to his own use; and one of such purchasers who conspired with the persons making the representations and received a part of the difference between the $25,000 and the $625, is liable to the other purchasers for such part so received by him, though he himself had no contractual relations with the other purchasers. Persons suing in such an action may waive the tort and sue the party making the representations for the balance of the money so obtained.

Reyer *v.* Blaisdell (Colorado Appeals), 143 Pacific, 385.

TRESPASS.

OWNERSHIP OF ORE WRONGFULLY MINED.

The description in the mortgage of a mining corporation reciting that all the real property of the mining corporation situated in the county and particularly described, also all lodes, mineral mining property, flumes, ditches, water rights, mill sites, and other mining properties that may thereafter be acquired by the corporation, by location, purchase, or in any other manner whatsoever, can not be extended beyond the import of the descriptive instrument and therefore a reasonable meaning and the intent of the parties can not be held to include ores wrongfully mined and taken from the mining property adjoining the property of the corporation executing the mortgage, for the purpose of holding the mortgagee as a trustee of the owner of the wrongfully mined ore.

Clinton Mining & Mineral Co. *v.* Trust Co., etc. (South Dakota), 151 Northwestern, 998, p. 999, April, 1915.

ORES WRONGFULLY MINED—EFFECT OF MORTGAGE.

Ores wrongfully mined from a mining claim by a mining corporation owning an adjoining claim and which had mortgaged all its property, including all lodes, mineral and mining property that may be acquired by location, purchase, or in any other manner whatsoever, can not be included within the terms of such mortgage for the purpose of making the mortgagee a trustee for the benefit of the owner of the ores so wrongfully mined.

Clinton Mining & Mineral Co. *v.* Trust Co., etc. (South Dakota), 151 Northwestern, 998, p. 999, April, 1915.

PLACER CLAIMS.

EFFECT OF PATENT ON KNOWN LODES.

A placer patent is, except in cases of fraud, a conclusive determination, made by a special tribunal having jurisdiction to determine, that the land conveyed contains no lodes or mineral lands, and the patent conveys the lands, though mineral or containing lodes, but by defeasible title if secured by fraud; but such title is valid against all the world save that when secured by fraud the United States by direct suit if brought within time may annul the patent and divest title.

Barnard Realty Co. *v.* Nolan, 215 Fed., 996, p. 1000.

PATENTED PLACER CLAIM—PROOF OF KNOWN LODE.

Proof to establish that a lode was known to exist when a placer patent was applied for must show that at that time the lode was clearly ascertained and defined and of such known extent and content that in view of all circumstances and conditions affecting its worth, such as the importance locally attached to like lodes under similar conditions, ease or difficulty of development, facilities for ore treatment, cost of mining and reducing ores, with the reasonable probabilities of development, it would then have justified location and development, and by reason of which it and the area attaching to, or excluded with it, were then more valuable than for placer mining purposes.

Clark-Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 963.

KNOWN LODE—WHAT CONSTITUTES.

Float, outcroppings, lodes, and abandoned locations, separately or combined, are not sufficient to constitute a "known lode" within the exclusion of the placer mining law; but to be impressed with such character the lode must, at the time of application for placer patent, be clearly ascertained and defined and of such extent and content

that it will then, in view of present conditions, justify development and exploitation, and because of which the placer claim is valuable and more valuable than for placer mining purposes. Subsequent development, however marvelous the results, is immaterial if the lode be not thus "known" when the application for the placer patent is made.

Barnard Realty Co. *v.* Nolan, 215 Fed., 996, p. 999.
Clark-Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 964.

KNOWN LODES—PLACER PATENT.

Neither a deed for a placer location nor a lode location upon the land by the placer applicant himself is prima facie evidence of a known lode, and both are deprived of all value by evidence of the nonexistence of a lode, and a lode subsequently located within the placer limits can not be "known to exist" at the time of a placer application for patent, where it appears that it was in fact discovered in the bedrock when the placer deposits were removed by extensive work long subsequent to the patent application.

Barnard Realty Co. *v.* Nolan, 215 Fed., 996, p. 998.

TITLE TO KNOWN LODES.

Known lodes, though unidentified and indefinite, are excepted and excluded from placer patents, and title to them remains in the United States, and at any time thereafter they may be, by strangers to the patent, possessed, located, and patented as any other lode upon public lands.

Barnard Realty Co. *v.* Nolan, 215 Fed., 996, p. 999.

KNOWN LODES—JUDGMENT QUIETING TITLE.

A decree in an action to quiet title to a placer location as against conflicting lode claims is not res judicata in respect to the United States and persons not parties, and such persons can relocate the lodes and relitigate the issue over and over, and if no title as to such known lodes passes by the placer patent it must remain wholly in the United States and neither laches nor limitations can vest title in the placer patentee.

Barnard Realty Co. *v.* Nolan, 215 Fed., 996, p. 1000.

APPLICATION FOR PLACER PATENT—OWNERSHIP OF KNOWN LODES.

Where lodes known to exist are excepted from a placer grant, title to them continues in the United States and they are open to location as lodes in public land by third persons at any time.

Clark Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 963.

APPLICATION FOR PLACER PATENT—KNOWN LODES EXCEPTED.

In an application for a placer patent the Land Department requires evidence as to the character of the land, and if the proof shows the existence of known lodes within a placer claim the applicant is required to survey them and if not claimed and included in his application he is required to exclude them and may then enter and pay for the net area of his placer claim and the patent conveys to him the net area alone; but if the proof shows there are no known lodes existing within the placer limits the applicant enters and pays for the entire area of his placer claim and patent issues covering the whole thereof. The law does not authorize the Land Department to insert in a patent an exception as to the existence of lodes within the placer limits broader than the law implies.

Clark-Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 963.

ANNUAL LABOR ON LODE CLAIMS WITHIN PLACER LIMITS.

Proof of the annual labor made in good faith on lode locations within placer limits is immaterial on the question of the knowledge of the existence of the lodes at the time of the placer application.

Clark-Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 964.

EFFECT OF PATENT FOR LODE ON PLACER CLAIM.

Subsequent patents for lode claims within the limits of a patented placer claim are immaterial on the question of the knowledge of the existence of such lodes at the time of the placer application, and the lode patents are not evidence of the known existence of such lodes at the time of the placer patent.

Clark-Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 965.

PATENTS.

PATENT FOR MINING CLAIMS—PRESUMPTIONS.

When a mining claim has been duly patented the conclusive presumption is that there was a discovery found within the limits of the patented claim, that the land was properly located and the boundaries of the claim so marked on the ground as to embrace not exceeding 300 feet on each side of the middle of the vein, and not exceeding 1,500 feet in length along the vein, and that all preliminary and precedent acts necessary to authorize and justify the issuance of the patent had been performed as the law required.

Stewart Mining Co. *v.* Bourne, 218 Fed., 327, p. 328.

PLACER PATENT—BURDEN OF PROOF AS TO LODES.

After the issuance of a patent for a placer claim a third person claiming the existence of a known lode within the placer limits has the burden of proving that such lode was known to exist when the placer patent was applied for and the proof must be clear and convincing, in quality and quantity that inspires confidence and produces conviction.

Clark-Montana Realty Co. *v.* Ferguson, 218 Fed., 959, p. 963.

STATUTES RELATING TO MINING OPERATIONS.

CONSTRUCTION, VALIDITY, AND EFFECT.

OHIO "RUN-OF-MINE" ACT.

The objects of the Ohio "run-of-mine" or antiscreen coal mine law of 1914 were to eliminate the objections to the "run-of-mine" basis of payment to miners and to enact a system fair alike to employer and miner. The first section provides for the payment of the miners according to the total weight of coal in a mine car as removed from the mine, with the percentage of impurities to be determined by the industrial commission of the State; and operators, under the provisions of the second section, are not obliged to compensate miners for everything sent out in mine cars, but the percentage of impurities as determined by the industrial commission is to be excluded from the calculation; and while the industrial commission is authorized to determine the percentage of impurities and enforce its orders relating thereto, yet the ascertainment of the commission is not a limitation upon the right of the operators and miners to agree upon deductions of their own arrangements as to the amount of slate, sulphur, rock, or dirt, and they may substitute their own agreement in that respect instead of that of the commission. The law does not prevent the operators from screening their coal as they see fit for other purposes and fit it for market in such wise as they may deem advisable; but the provision for screening is for the purpose only of calculating the amount to be paid miners for mining coal. The statute does not make the orders of the industrial commission final or conclusive, but makes them only prima facie reasonable, and operators are entitled, upon petition, to a hearing upon the reasonableness of any order and are given the right to bring an action in the supreme court to test the reasonableness or validity of any such order. The statute is not unconstitutional as unduly abridging the freedom of contract in prescribing the particular method of compensation to be paid by operators to miners for the production of coal, under the constitution of Ohio,

which provides that laws may be passed "for the regulation of methods of mining, weighing, measuring, and marketing coal, oil, gas, and other minerals."

Rail & River Co. *v.* Yaple (Ohio Industrial Commission), 236 U. S., 338, p. 345.

STATUTORY PROVISION NOT VARIED BY A CUSTOM.

While the duties and liabilities of a mine operator and a miner may sometimes depend upon a custom of the mine where there is no statute covering the subject, yet where a statute speaks on the subject the terms of the statute can not be modified by proof of a custom contravening the statute; and accordingly where a statute imposes upon the mine operator the duty of furnishing props and caps only after the miner has selected and marked the same an injured miner in an action for damages can not prove a custom of the mine by which the mine operator furnished the props and caps without such marking, as a custom of the mine contrary to the provisions of the statute is void, the very purpose of the statute being to do away with uncertain conditions and to prescribe with reasonable certainty the duties and liabilities of the mine owner and of the miner.

Palmer *v.* Empire Coal Co. (Kentucky), 172 Southwestern, 97, p. 98, January, 1915.

STATUTE—WORDS CHANGED.

In section 2739b of the Kentucky statutes, requiring mine operators to provide and furnish the miners a sufficient number of caps and props and requiring the miners to keep the roof propped, the word "worked" should be "marked" and the words "the same" following refer to the caps and props, so that the latter clause of the statute would read, "it is the duty of said miners to keep the roof propped, after the miner has selected and marked the caps and props."

Palmer *v.* Empire Coal Co. (Kentucky), 172 Southwestern, 97, p. 98, January, 1915.

COMMON-LAW RIGHTS ABOLISHED.

The mining statute of West Virginia transfers liability which the common law would impose upon the operator of a mine to the shoulders of the mine foreman and abrogates the common-law right of action of the injured miner against the mine operator; and this imposition of liability upon the mine foreman is in derogation of his common-law right of service without it, and the abrogation of the injured servant's right of action against the mine operator is also in derogation of his common-law right as an employee, and the statute accordingly ought not and can not have a construction and effect beyond that clearly indicated by its terms.

Crockett *v.* Black Wolf Coal & Coke Co. (West Virginia), 83 Southeastern, 987, p. 988, December, 1914.

IMMUNITY AFFORDED OPERATOR.

The immunity of employers of labor in coal mines, given by the statute providing for the appointment of mine foremen and prescribing their duties, is limited to the duties so imposed upon each mine foreman in express terms or by clear implication arising out of the terms of the statute; but it does not extend generally to omissions or failure to exercise reasonable care and diligence to provide and maintain suitable and safe machinery, tools, and appliances for use by the miners as instrumentalities in the performance of their duties.

Crockett *v.* Black Wolf Coal & Coke Co. (West Virginia), 83 Southeastern, 987, p. 988, December, 1914.

REGULATING MINES—POLICE REGULATIONS.

The statute of Kansas, chapter 222, Laws of 1911, is not unconstitutional and invalid because compliance with the act involves a greater expense to mining companies than do other statutes relating to mining operations, where there is no showing that the requirements are confiscatory or unreasonable in consideration of the object to be attained; nor is it unconstitutional because the act is discriminatory in that it places burdens upon coal-mine operators, while the operators of lode, zinc, gypsum, and salt mines are free from such burdens.

State *v.* Reaser (Kansas), 145 Pacific, 838, p. 839, January, 1915.

WASHHOUSES.

Chapter 222, Kansas Laws of 1911, requiring operators of coal mines to maintain washhouses in connection with their mines is a police regulation, and the determination of the necessity and wisdom of a police regulation rests with the legislature, and if there is reasonable grounds for exercising such power the courts should not interfere, and the particluar act in question contemplated the health and comfort of the employees, and the framers of the law will be presumed to have been possessed of such general knowledge and to have made such special investigations of the conditions at coal mines as to them was deemed necessary to justify the enactment of the statute.

State *v.* Reaser (Kansas), 145 Pacific, 838, p. 839, January, 1915.

GUARDING SHAFTS AND DRIFTS—DITCHES NOT INCLUDED.

The statute of Montana declaring a penalty for sinking any shaft or running any drift or cut without guarding it does not include a trench or sewer in a city for the purpose of laying sewer pipe.

McLaughlin *v.* Bardsen (Montana), 145 Pacific, 956, January, 1915.

RAILROAD RIGHT OF WAY—TRAM ROAD NOT A RAILROAD.

A tramway built by a railroad company along a steep mountain side upon a steep grade with 12-pound rails, with only 20 inches between the rails, laid on ties 4 by 4 inches and 4 feet in length, not intended for use as an ordinary steam railroad, but intended to be used and in fact leased to and operated by mine owners for hauling ore from and supplies to its mines from and to steam railroad connections, a distance of less than 3 miles, the only cars used being ore cars of 2 to 3 tons' capacity drawn up the grade by horsepower and returned by gravity, is not a railroad within the meaning of the act of Congress entitled "An act granting to railroads the right of way over the public lands of the United States," approved March 3, 1875 (18 Stat., 482), so as to entitle the railroad company to claim 100 feet of ground on either side of the center of such tramway.

Denver & Rio Grande Railroad Co. *v.* Bolognese (Utah), 143 Pacific, 129, p. 132.

STATUTE REQUIRING CASH PAYMENT OF WAGES.

The statute of Tennessee, Acts 1913, chapter 29, making it a misdemeanor punishable by fine if a coal company or other corporation operating a supply store in connection with its business fails to pay in cash its employees at stated periods the wages due them, is in violation of the constitutional provision to the effect that the legislature shall pass no law authorizing imprisonment for debt in civil cases, as the act while not directly authorizing imprisonment for debt, does attempt to create a crime for the failure to pay in cash, and for such crime provides a penalty, which may or may not be followed by imprisonment, and is an indirect imposition of imprisonment for the nonpayment of debt, and is therefore clearly within the constitutional inhibition.

State *v.* Pend (Tennessee), 170 Southwestern, 56, October, 1914.

STATUTE PROHIBITING THE ISSUE OF SCRIP IN PAYMENT FOR SERVICES.

The statute of West Virginia prohibiting under penalty any corporation, company, or person from issuing or giving to any employee any scrip, token, draft, check, or other evidence of indebtedness, payable or redeemable otherwise than in lawful money, and providing that any such scrip, draft, check, or other evidence of indebtedness shall be taken and held to be a promise to pay the sum specified therein in lawful money, is not in violation of the Federal Constitution and does not impose undue restrictions upon the liberty of contract and does not violate any constitutional provision and is not an illigitimate exercise of the State's police power; but the statute is constitutional and valid.

Atkins *v.* Grey Eagle Coal Co. (West Virginia), 84 Southeastern, 906, p. 907, March, 1915.

INSPECTION OF ILLUMINANTS USED IN MINES.

The Alabama statute of 1911, Acts of 1911, p. 568, entitled an act "to regulate the inspection and use of illuminants in mines in the State of Alabama and sales of illuminants for use in mines," is not in conflict with section 77 of the constitution of Alabama which provides that no State office shall be continued or created for the inspection or measuring of any merchandise or commodity, but permits any county or municipality to appoint such officers when authorized by law, as this constitutional provision was not designed to prevent the conferring upon and the exercise by State officers all inspecting powers, but to prevent the legislature from creating and continuing any State office for that purpose and it was not the purpose of the provision to prevent the inspection of commodities but to prevent the creation of offices solely for that purpose, where the statute does not create any office whatever but provides for the inspection of illuminants by the State mine inspector, an office created long before the enactment under consideration.

Wofford Oil Co. *v.* Burgin (Alabama), 66 Southern, 931, p. 932, December, 1914.
Burgin, Ex parte (Alabama), 68 Southern, 49, January, 1915.

REPEAL OF INSPECTION ACT.

The statute of Alabama, Acts of 1911, page 568, entitled an act "To regulate the inspection and use of illuminants in mines in the State of Alabama and sales of illuminants for use in mines," revises the whole subject matter of the local act of 1901, page 1249, and must be construed as intending to set up a new system of inspection and as being designed as a substitute of the former act, and accordingly repeals the same.

Wofford Oil Co. *v.* Burgin (Alabama), 66 Southern, 931, p. 932, December, 1914.
Burgin, Ex parte (Alabama), 68 Southern, 49, January, 1915.

REGULATION OF NATURAL-GAS RIGHTS.

The fixing of rates to be charged by public-service corporations to their customers in West Virginia is not an unlawful regulation of insterstate commerce, as the regulation of companies engaged in the transportation of gas is expressly excluded from the scope of the interstate commerce statute; but neither the statute of West Virginia nor the orders of the public-service commission purport to interfere in any manner with the transportation of natural gas from that State to other States, and the statute does nothing except regulate prices of natural gas to the citizens of West Virginia, charged by corporations operating in that State under State authority; but the interflow of gas from Ohio and Pennsylvania into West Virginia, or out of West Virginia into Ohio and Pennsylvania, according to

the pressure from the main gas pipes, as common reservoirs, can not affect the power of the State of West Virginia to make reasonable regulations as to rates for gas furnished to its own citizens.

Manufacturers' Light & Heat Co. *v.* Ott, 215 Fed., 940, p. 944.

PUBLIC-SERVICE COMMISSION—DUTY AS TO OIL AND GAS.

In the public-service commission of the State of West Virginia the legislature of the State has not delegated its legislative power, but merely provided an agency for carrying out the legislative scheme with reference to public-service corporations, and the legislature may require of such commission the application of general rules to particular situations and the investigation of facts, with a view to making orders in a particular matter.

Manufacturers' Light & Heat Co. *v.* Ott, 215 Fed., 940, p. 943.

PUBLIC-SERVICE COMMISSION—GAS COMPANIES.

The term "gas companies" used in the public-service commission act of West Virginia, embraces companies furnishing natural gas, and the statute was enacted in view of the fact that a very large part of the gas consumed in the State was natural gas.

Manufacturers' Light & Heat Co. *v.* Ott, 215 Fed. 940, p. 944.

INTERSTATE SHIPMENT OF OIL—INSPECTION—INJUNCTION.

An injunction will lie under the statute of North Dakota for the inspection of oil to prevent the State oil inspector from holding up oil in transit from other States into North Dakota, for nonpayment of inspection fees, where such fees are materially in excess of the amount necessary to pay the expenses of inspection.

Bartel's Northern Oil Co. *v.* Jackman (North Dakota), 150 Northwestern, 576, p. 579, January, 1915.

INSPECTION OF OIL IN TRANSIT—FEES INVALID.

The statute of North Dakota providing for the inspection of oil during transit, brought from other States into North Dakota, and fixing the fee for such inspection materially greater than the cost thereof, becomes not only a police measure but a revenue measure, and to the extent that the fees exceed the reasonable necessary cost of inspection, the tax is invalid, as such statute is in conflict with the commerce provisions of the Federal Constitution.

Bartel's Northern Oil Co. *v.* Jackman (North Dakota), 150 Northwestern, 576, p. 580, January, 1915.

PERMITTING OIL TO ESCAPE—PROTECTION OF STATUTE.

A statute of Indiana, section 9062, Burn's Annotated Statutes, 1914, requires oil and gas produced from wells to be safely and securely confined in wells, pipes, or other safe and proper receptacles, and the purpose of the statute is to require persons producing live oil from wells securely and safely to confine it in receptacles on their premises to prevent its escape; and a person operating a dredge in the construction of a public ditch or drain is entitled to the benefit of the provisions of this act; and an oil-producing corporation that permitted its oil to escape from a tank where it was stored, and flow down into such ditch, against and around the dredge, is liable to the owner of such dredge where it was destroyed by reason of the oil accidently taking fire from the fire on the dredge, even in the absence of any question or allegation of negligence.

Rock Oil Co. *v.* Brumbaugh (Indiana Appeals), 108 Northeastern, 260, p. 263, March, 1915.

See Commercial Union Assurance Co. *v.* Gulf Refining Co. (Texas Civil Appeals), 174 Southwestern, 874, March, 1915.

OPERATION OF WORKMEN'S COMPENSATION LAW—ALLOWANCE FOR INJURY.

In an action by a miner under the workmen's compensation act of Kansas (Laws 1911, chap. 218; amended, Laws 1913, chap. 216), where the evidence showed that the miner's arm was broken and the freedom of its movement was permanently impaired, restricting his activity in the occupation in which he was engaged and in which he was earning $17.50 a week, the court found the plaintiff had suffered a total disability for six months following the accident and allowed him $210 on that account, with an additional allowance of $1,014 based upon the finding of partial disability for a period of six and one-half years; and under the statute it is within the discretion of the trial court to make the judgment for a lump sum or for periodical payments, and where there is no abuse of this discretion the judgment will not be interfered with on appeal.

Cain *v.* National Zinc Co. (Kansas), 148 Pacific, 1165, March, 1915.

ILLINOIS MINING ACT—EFFECT OF WORKMEN'S COMPENSATION ACT.

Section 1 of the compensation act of Illinois makes it optional with both the coal-mine operator and the miner whether or not they will accept the provisions of the act, and the application of such act being optional with either or both parties, the compensation act did not repeal the mining act and does not deprive an injured miner of a right of action under the Illinois mining act for injuries sustained by reason of the failure of a coal-mine operator to comply with the provisions of the mining act.

Eldorado Coal & Mining Co. *v.* Mariotti, 215 Fed., 51, p. 55.

STATUTORY RIGHTS AND DUTIES.

MINING RIGHTS ACQUIRED UNDER STATUTE—PROTECTION.

The statute of Missouri, section 8409, Revised Statutes of 1909, provides that when an owner or lessee of real estate permits another to enter and dig for lead ore or other minerals thereon with his consent, and such persons having in good faith dug or opened any shaft, mine, or deposit of mineral, or extended or opened from any shaft or mine any room, drift, or entry, such person shall have the exclusive right as against the owner to continue the work and mine and dig such shaft or deposit of mineral, and the statute protects a person, as against an action of ejectment by the landowner, who with the knowledge of such owner, and under an agreement with the lessee of the original owner, entered upon a part of the leased premises and engaged in mining in good faith, opened shafts and extended and operated shifts therefrom and assembled machinery for carrying on mining operations, as such person acquired a mining right under the statute and is not a trespasser, though the lessee on whose consent and agreement such person entered upon the mining property, forfeited the lease.

G. M. Mining Co. *v.* Hodge (Missouri Appeals), 170 Southwestern, 689, November, 1914.

DUTIES IMPOSED ON OPERATOR.

DUTY TO FURNISH PROPS—HEADERS INCLUDED IN STATUTE.

The statute of Kentucky, section 2739b, subsection 7, requires the owner or operator of every mine to provide and furnish to the miners a sufficient number of caps and props. The props referred to are upright posts wedged between the roof and the flooring to support the roof, while a cap is a square piece of plank or block wedged between the top of the posts and the roof to better hold the roof, while "headers" are longer pieces of plank extending over more of the roof and supported by two props, one at each end, and by the use of the header a larger area of roof can be held secure with fewer props, and in this way serves the purpose of a cap; the use of headers being customary and frequently necessary, they are necessarily embraced within the materials named in the statute; and a mine operator may be liable for failure to furnish a header when requested by a miner as for a violation of the statute.

Big Branch Coal Co. *v.* Wrenchie (Kentucky), 170 Southwestern, 14, p. 16, November, 1914.

PROVIDING SAFE PLACE.

One of the positive duties which the statute of Oklahoma imposes upon a coal-mine operator is to exercise care to provide a reasonably safe place for his miners to work, and the degree of care demanded of

him depends upon the natural dangers incident to the employment; and this means that a greater degree of care is required of a master who is a coal operator working his servants in a mine underground, than of a master who requires his servants to work in a less hazardous occupation out in the open; and the failure of a mine owner to discharge this positive duty constitutes primary negligence and renders him liable for any injury resulting therefrom.

Rock Island Coal Mining Co. *v.* Davis (Oklahoma), 144 Pacific, 600, p. 605, December, 1914.

The mine-foreman statute of West Virginia does not absolve the mine owner or operator from his common-law duty to exercise reasonable care to provide reasonably safe machinery, tools, and appliances for use in the mine, and make the mine a reasonably safe place for work, except in so far as this duty is devolved upon the mine foreman; nor does it absolve him from liability for injury resulting to a miner in the mine from the operator's failure to make such provision, or his provision of defective or unsafe appliances, or his failure of duty as to the safety of the mine as a place of work in those instances in which such duty is not cast upon the mine foreman.

Crockett *v.* Black Wolf Coal & Coke Co. (West Virginia), 83 Southeastern, 987, p. 988, December, 1914.

DUTY TO TIMBER MINE.

The statute of Oklahoma makes it the positive duty that the owner of a mine owes to his miners, after the mine is opened and timbered, to use reasonable care and diligence to see that the timbers are properly set and to keep them in proper condition and repair, and for this purpose he is required to provide a competent mine boss or foreman, to make timely inspection of the timbers as well as the roofs of the mine, to the end that the miners may not be injured by defects or dangers which a competent mining boss or foreman would discover and remove.

Rock Island Coal Mining Co. *v.* Davis (Oklahoma), 144 Pacific, 600, p. 605, December, 1914.

DUTY TO FURNISH SAFE APPLIANCES—TROLLEY WIRES.

The mining statute of West Virginia does not place the duty of inspection, oversight, and repair of the electrical trolley wires used to communicate power for haulage purposes in the mine upon the mine foreman and its terms are not broad enough to fix it upon him, in respect to the miner making actual use of such appliance, in the discharge of the duties he was performing under his contract of employment. The trolley wire used for the purpose of communicating power is an implement used in the miner's work, and the statute does not say that the mine foreman shall see that the miners are pro-

vided with safe or suitable tools and appliances, and accordingly the duty of maintenance of the safety of the trolley wire for such purpose was not devolved upon the mine foreman. The mine operator is therefore liable for the death of a miner caused by a defective trolley wire and the duty devolves upon him to inspect such wire and other similar machinery and appliances, as a matter of precaution for the safety of his miners; and in view of the dangerous character of trolley wires, the law requires the exercise of a high degree of care and diligence in the matter of inspction as well as other duties respecting it.

Crockett *v.* Black Wolf Coal & Coke Co. (West Virginia), 83 Southeastern, 987, p. 989, December, 1914.

OPERATOR TO FURNISH PROPS—DUTY TO PROP.

Subsection 7 of section 2739b, Kentucky Statutes, provides that the operator of every mine shall furnish to miners a sufficient number of caps and props to be used by miners in securing the roofs in their rooms and such other working place whether by law or custom of those engaged in such employment it is the duty of the miners to keep the roof propped; but does not by its terms impose the duty of propping the roof upon the miner or mine operator; but this duty may, by agreement or custom, be imposed upon either the miner or mine owner and proof of such custom is admissible, and it is also proper to prove that the United Mine Workers of America, a labor organization, had an agreement with the mine operator by which it was the duty of the machine operator to direct his "loaders" to prop the roof and if they did not do so and it became dangerous, the miner should thereupon report it to the boss.

Old Dominion Coal Co. *v.* Denney (Kentucky), 169 Southwestern, 1016, p. 1017, October, 1914.
Borderland Coal Co. *v.* Small (Kentucky), 170 Southwestern, 8, p. 9, November, 1914.

DUTY TO PROVIDE SAFE APPLIANCES—DERAILING SWITCH.

A complaint is sufficient if it charges that the defect in the condition of the ways, works, machinery, or plant consisted in the negligent failure of the operator to provide or maintain a dead latch or derailing switch to derail the cars on becoming loose on the incline in the mine and by reason of which a trip of cars, being hauled up the slope, became loose and ran back down the slope upon the miner.

Burnwell Coal Co. *v.* Setzer (Alabama), 67 Southern, 604, p. 605, December, 1914.

DUTY TO FURNISH SAFE HAULAGEWAYS.

Under the general provisions of the Pennsylvania statutes placing the workings of a mine under the mine foreman's charge and supervision, and under certain special provisions to be found in the stat-

utes, he is responsible for all work in the course of the construction of passageways and for their future proper support, for the effect of all temporary delay in operations therein and for the proper maintenance of the appliances used in such operations; but where a tunnel or haulageway has actually been constructed and the track laid for such a considerable length of time that the passageway, with its equipment, has become a part of the regular and established plan of the mine, and the owner or operator, through the superintendent, has knowledge of the conditions, when such conditions are more hazardous than the surroundings require, even though they do not present an immediate danger, if it reasonably can be foreseen that in the operation of the mine they are such as may well result in injuries to miners, then it is the duty of the owner or operator to have them changed and made more safe, and the failure so to do is negligence that will give rise to a liability on the part of the operator.

Watson *v.* Monongahela River Consolidated Coal & Coke Co. (Pennsylvania), 93 Atlantic, 625, p. 627, January, 1915.

DUTY TO MAINTAIN PASSAGEWAYS.

The act of 1891 of Pennsylvania provides that all passageways in mines shall be made of sufficient width to permit persons to pass moving cars in safety, and provides that mine owners shall place the underground workings, and all that is related to the same, under the charge and daily supervision of a competent person who shall be called "mine foreman;" and the statute defines "workings" as embracing all the excavated parts of a mine, which necessarily includes passageways. The act of 1893 contains provisions concerning the mine foreman and his duties which indicate that passageways are under his general control, and this statute fixes him with certain express duties in connection therewith; but there is nothing aside from the part of the act which deals with the width of entries at places where spragging is necessary and also providing that where there is a space 4 feet between the car and rib, it shall be sufficient for shelter; but these do not show a legislative intent to prescribe a statutory width for passageways and accordingly in bituminous mines the common-law obligation is upon the owner or operator to have a proper width in all established passageways, according to the necessities of each particular place, and this obligation is to the same extent as though the duty had been expressed in the act of 1893 as in the act of 1891.

Watson *v.* Monongahela River Consolidated Coal & Coke Co. (Pennsylvania), 93 Atlantic, 625, p. 627, January, 1915.

DUTY TO MAINTAIN SAFE PLACE—EXCEPTION.

The rule that an employer must exercise reasonable care to furnish his employees a reasonably safe place in which to work does not apply to the operator of a quarry and does not render him liable for negligence in failing to exercise such care where the very work which the employee is employed in doing is of such a nature that its progress constantly produces changes in the conditions and surroundings, and consequently the hazards arising therefrom, to which the employee is exposed, are regarded as the ordinary dangers of his employment and as such assumed by him.

Losasso *v.* Jones Brothers' Co. (Vermont), 93 Atlantic, 266, p. 269, February, 1915.

INSPECTION AND VENTILATION—QUESTION OF FACT.

Section 4986, General Statutes of Kansas, 1909, requires all mines generating fire damp to be kept free from standing gas and that every working place shall be carefully examined every morning with a safety lamp by a competent person before any miner is allowed to enter therein. Section 5005, General Statutes of Kansas, 1909, makes similar requirements and provides also that the hydrogen or fire damp generated in working places must be diluted and rendered harmless before miners are permitted to enter such places with a naked light. The manifest object of these sections was to prohibit miners from injuries as well as from fatalities, and it is a well-recognized fact that while mines containing "horsebacks" often manifest the presence of gas in a comparatively slight degree, nevertheless in other instances gas is present in such amounts that ignition would mean serious injury, and in such cases it is a question of fact to be determined by the jury as to whether or not a mine operator knows that the mine is generating fire damp or has been generating it in such quantities as to make a reasonable person know that it was doing so.

Ward *v.* Mackie Fuel Co. (Kansas), 146 Pacific, 1138, March, 1915.

DUTIES IMPOSED ON MINER.

REQUEST FOR PROPS—MAKING WORKING PLACE SAFE.

Section 2489–16a of the Code Supplement of 1913 of Iowa imposes upon each miner in a mine the duty to examine his working place and prohibits him from commencing to mine or load coal or other material until such place is made safe, and requires each miner to securely prop and timber the roof of his working place and requires him to prop or timber any draw slate or other like material before working under the same and imposes upon him the duty of making a request on the operator for suitable timbers and props; and these

duties are continuing duties and rest upon the miner regardless of whether the operator has supplied him with suitable props or not, and he must not permit his attention to be diverted to any other work until he has made the examination and inspection required; and he is positively forbidden to commence to mine or load coal until his working place is made safe; but this statute does not make the miner an insurer of the safety of the mine, regardless of any breach of duty on the part of the operator. The term "safe," as used with reference to the miner's working place, is necessarily a relative one, and by its very meaning involves degrees, and the miner is not required to keep out of his room because it is not propped, but he is required to inspect it promptly and diligently and to discover its dangers to the extent that diligence will discover the same; and in the performance of this duty he is necessarily exposed to danger and if he discovers any indications of its condition which a reasonably prudent man would not regard as safe, then his working place is not safe within the meaning of the statute; but the danger in such case may be potential only and in the judgment of the miner should be propped up before it becomes imminent, and the failure of the miner to prop the place of work, where the danger is not imminent, will be excused where the operator fails to furnish the props and timbers when requested where it may have been beyond the range of ordinary diligence, skill, and experience to have discovered the danger.

Edgren *v.* Scandia Coal. Co. (Iowa), 151 Northwestern, 519, p. 521, March, 1915.

KEEPING WORKING PLACE SAFE.

Where the condition in a mine, from which the injury to a miner suing for damages is alleged to have resulted, was the immediate product of the progress of the work in which the miner was properly engaged, such condition could not have been a defect within subdivision 1 of section 3910 of the statute of Alabama.

Sloss-Sheffield Steel & Iron Co. *v.* Terry (Alabama), 67 Southern, 678, p. 680, December, 1914.

OPERATOR'S FAILURE TO COMPLY WITH STATUTORY REGULATIONS.

PROXIMATE CAUSE OF INJURY.

A violation of the statute of Colorado by a coal company in failing to equip its mine with a speaking tube or telephone connection as required by the statute, does not constitute actionable negligence and render it liable for the death of a miner, where the failure to so equip the mine with speaking tubes or telephones in no way contributed and was not the proximate cause of the miner's death.

Colorado Capitol Coal Mining Co. *v.* Chatfield (Colorado) 143 Pacific 1095, p. 1096.

LIABILITY OF OPERATOR—PLEADING.

A complaint in an action by an injured miner under the statute of Alabama is sufficient where in the usual terms it avers the miner's injuries and damages to have been proximately caused by reason of the negligence of a superintendent or superior in the service and employment of the operator, to whose orders or directions the injured miner was bound to conform and did conform, which negligence consisted in this, that the said superintendent or superior negligently allowed the work of the injured miner to be performed in a manner dangerous to the safety of the miner and negligently ordered the miner to place a plank or piece of timber upon which steam shovels or wheels were to run, and in conforming to the order a wall of ore and clay near which the miner was engaged in the performance of his duty at the time of the injury was insecure and unsafe, so that a large embankment therefrom fell upon the miner, injuring him.

Sloss-Sheffield Steel & Iron Co. *v.* Terry (Alabama), 67 Southern, 678, p. 680, December, 1914.

FAILURE TO COMPLY—PROXIMATE CAUSE.

The mere noncompliance with a positive statute on the part of a mine operator or manager is not sufficient to entitle a miner to recover damages for injuries sustained, unless the noncompliance with the statute is the proximate cause of the injury, and unless a compliance with the statute would have avoided the accident and saved the miner from injury.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 911, December, 1914.

OPERATOR'S DISREGARD OF STATUTE.

A miner by using the means provided by a mine operator or manager is not guilty of contributory negligence, and mere contributory negligence in this respect on the part of the miner will not defeat a right of recovery where he is injured by the willful disregard of a positive statute, either by an act of omission or commission on the part of the mine operator or manager.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 910, December, 1914.

FAILURE TO GUARD MACHINERY—PROXIMATE CAUSE.

A coal-mining plant is an establishment within the meaning of the statute of West Virginia requiring operators in manufacturing, mechanical, and other establishments where machinery, belting, shafting, gearing, drums, and elevators are so arranged and placed as to be dangerous to persons employed, to safely and securely guard such machinery when possible, and if not possible, then notice of the danger must be conspicuously posted. This statute was intended to

guard against injuries to employees by coming in direct contact with machinery, belting, shafting, gearing, drums, and elevators in the establishment, but does not apply to injuries arising indirectly from any such conditions; and a miner injured by material falling down the shaft of a mine can not recover as for an alleged violation of this statute on the ground that the cage used in lowering and hoisting materials in the mine was not properly secured, as the statute does not provide that mine hoists or elevators shall be guarded when carrying coal and other materials to keep them from falling inside the shaft; and if there is a liability on the part of the mine operator for negligence it must be enforced under common-law rules and not under this statute.

Ferguson *v.* Middle States Coal & Coke Co. (West Virginia), 84 Southeastern, 573, p. 574, February, 1915.

FAILURE TO COVER CAGE—APPLICATION OF STATUTE.

Proof that a mining company owning coal lands caused two shafts some 200 feet apart to be sunk to and through the vein of coal to a depth of about 540 feet and thereafter employed underground a force of more than 20 miners, causing entries to be driven in every direction, constructing roadways and air courses, all such air courses and entries being of a length of more than 2,000 feet, laying tracks and switches, mining and taking out about 2,000 tons of coal, is sufficient to show a mine and to bring the plant and mine within the operation of the Illinois mines and miners act and to render the operator liable for the death of a miner caused by falling material because of the failure of the operator to equip the cage as required by the statute with a proper and sufficient covering to protect miners riding thereon.

Hakanson *v.* La Salle County Carbon Coal Co. (Illinois), 106 Northeastern, 617, p. 618, October, 1914.

FAILURE TO USE CAGE IN SHAFT—VIOLATION OF STATUTE.

Section 6799, Revised Laws of Nevada, makes it unlawful for any person or corporation to sink or work through any vertical shaft at a greater depth than 350 feet unless the shaft shall be provided with an iron-bonneted safety cage to be used in lowering and hoisting miners, and the statute is not complied with by simply having such a cage upon the mining premises but not in actual use; and aside from this, the statute makes it unlawful to sink or work through any vertical shaft at a greater depth than 350 feet, unless in the lowering and hoisting of the miners, in conducting such work or such sinking, the shaft be provided with an iron-bonneted safety cage. A bucket and crosshead used in a vertical shaft in a mine at a greater depth than 350 feet is not a compliance with the act.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 909, December, 1914.

FAILURE TO USE SAFETY CAGE—DEFENSE.

The statute of Nevada requires a coal-mine operator working through any vertical shaft at a greater depth than 350 feet to use an iron-bonneted safety cage in lowering and hoisting miners, and it is no defense in an action for the violation of the statute to say that the miners failed to demand such a compliance to be used in lowering and hoisting them through the shaft, as the statute was not enacted with its primal object, that of punishment for its violation, but the penalty imposed for its violation was rather prescribed as a reminder that the law is a police regulation, enacted for the purpose of minimizing accidents which entail suffering, privation, and death on those who may be the unfortunate victims.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 909, December, 1914.

FAILURE TO WET COAL DUST—EXPLOSION.

An action for damages for the death of a miner caused by a violation of section 4380, Compiled Laws of Oklahoma (1909), was prosecuted upon the theory that the coal operator had violated the statute in negligently permitting coal dust to mingle with and clog the air in the mine, and that a windy shot coming in contact with such coal dust ignited it, causing the explosion that resulted in the death of the miner; and the action was defended upon the theory that the operator had complied with the law in wetting the accumulated coal dust, but the explosion was caused by a "windy shot" igniting the coal dust produced and suspended in the air by a "follow shot." A verdict and judgment for the plaintiff in such action can not be disturbed on appeal where the evidence reasonably tended to establish that at the time of the explosion the coal operator had violated the statute in the particular alleged, and where the evidence also was sufficient to show causal connection between the violation of the statute and the death of the miner.

San Bois Coal Co. *v.* Resetz (Oklahoma), 143 Pacific, 46, p. 48.

FAILURE TO VENTILATE MINE—QUESTION OF FACT.

The Pennsylvania mining law of 1891 requires the owner or operator of every mine to maintain an adequate supply of pure air for the same and to conduct the ventilating currents along the face of every working place in quantities sufficient to render harmless noxious or dangerous gases; to close with suitable material all crosscuts when necessary to close them permanently; to have automatically-closing doors; to use every precaution to insure the safety of the miners; and to use no light or fire therein except safety lamps where there is likely to be an accumulation of explosive gas, and makes the owner or oper-

ator liable for an injury occasioned by any violation of the act; and the fact that a miner testified that upon lighting a squib to fire a blast he was knocked down by an explosion of gas was not so incredible, nor was the cause of the explosion so disconnected with the employer's failure to ventilate the mine, as required by the above statute, as to justify a direction of a verdict by the court; but under such circumstances the question of the failure to properly ventilate the mine, as well as the cause of the explosion, should be submitted to the jury.

Delaware, etc., R. Co. *v.* Yurkonis, 220 Fed., 429, p. 433, January, 1915.

USING BETTER APPLIANCE THAN STATUTORY REGULATIONS.

Failure to use a statutory device or implement or appliance in the operation of a mine might not be a violation of the statute requiring the use of a particular device or appliance if the mine operator uses a device or appliance generally and customarily regarded as being better or more liable to insure safety than that provided by the statute.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 910, December, 1914.

VIOLATION OF STATUTE AS EVIDENCE OF NEGLIGENCE.

Whenever an act is enjoined or prohibited by statute and the violation of the statute is made a misdemeanor, any injury to the person of another caused by such violation is the subject of an action for the recovery of damages, and the violation of the statute is the basis of the right to recover and constitutes negligence per se.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 910, December, 1914.

IGNORANCE AND INADVERTENCE NO DEFENSE.

An inadvertent or ignorant failure to comply with the provisions of a statute in the operation of a mine is the same as an intentional evasion thereof, and neither can be proved as a defense in the prosecution for a violation of such statute.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 910, December, 1914.

LIABILITY FOR NONCOMPLIANCE.

Any conscious omission or failure of a mine operator to comply with a statute which requires that he furnish certain reasonable appliances for the protection of life and limb of the miners employed by him renders him liable for ensuing injuries.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 911.

FAILURE TO FURNISH PROPS—PROXIMATE CAUSE OF INJURY.

Section 2489–5a of the Code Supplement of 1913 of Iowa requires the owner or operator of a mine to keep a sufficient supply of caps and timbers to be used as props convenient and ready for use, and requires them to be sent in and delivered to the places where needed on request; and this statutory duty imposed upon the operator is predicated primarily upon the potential danger of an unpropped roof, a danger not necessarily imminent; but the operator is charged with the knowledge that the potential danger of an unpropped roof may become the imminent danger at any moment, and may occur in a way against which the reasonable diligence of the miner can not provide without props, and in a way to prevent discovery even by the reasonable diligence of the miner; and for a court to hold that a failure of an operator to comply with this provision of the statute is too remote to be the proximate cause of an ultimate injury of a miner would be to destroy the manifest purpose of the statute; and the violation of the statute at this point by the operator may be regarded as an obstacle in the way of propping the mine, and the failure of the miner to prop in such case should be deemed the failure of the operator, and hence if the failure to prop the mine is the proximate cause of an accident, then the negligence of the operator could be such proximate cause.

Edgren *v.* Scandia Coal Co. (Iowa), 151 Northwestern, 519, p. 521–523, March, 1915.

EFFECT ON CONTRIBUTORY NEGLIGENCE.

WHAT CONSTITUTES CONTRIBUTORY NEGLIGENCE.

The failure of a mine operator to perform the statutory duty causes the liability for all consequences to fall upon him, unless a miner, complaining of an injury from such failure, could see or know by ordinary care that the situation was imminently dangerous, and in such case there is no assumption of risk by the miner where the operator neglects a statutory duty, though the miner may still be guilty of contributory negligence such as will defeat a recovery; but in order to constitute contributory negligence there must be some act or failure on the part of the miner, in addition to the ordinary risks imposed by the character of his work under the conditions created by the operator's conduct, which would amount to culpable negligence on the miner's part, examples of which may be found in his failure to look, to observe, to test in some way the safety of the roof, or if it is unsafe and obviously so, and the danger thereby imminent, his continuing to work under such conditions.

Log Mountain Coal Co. *v.* Crunkleton (Kentucky), 169 Southwestern, 692, p. 694, October, 1914.

Jellico Coal Mining Co. *v.* Walls (Kentucky), 170 Southwestern, 19, p. 20.

FREEDOM FROM CONTRIBUTORY NEGLIGENCE.

A miner is not guilty of contributory negligence in working in an entry, although known to him to be defectively propped; nor by such working does he assume the additional risk of such defect, unless either it was his duty to remedy the defect, or unless in the course of the work he was to do, further propping was made, necessary, which it was his duty to do, and he negligently failed to do such further propping, and such negligent failure either solely caused the injury or contributed with such defect in producing it.

Lookout Fuel Co. *v.* Phillips (Alabama), 66 Southern, 946, p. 949, November, 1914.

DEFENSE OF CONTRIBUTORY NEGLIGENCE ABROGATED—PLEADING.

A complaint in an action by a miner for damages for injuries predicated on subdivision 1 of the employers' liability statute of Alabama (code section 3910) averred that the miner while in the service and employment of the defendant operator as a coal miner in one of its mines was injured by a fall of stone or slate from the roof or top of the entry in which he was at the time engaged in work under his employment, and that the injury so received was the proximate consequence of a defect in the condition of the ways, works, machinery, or plant of the operator in that the roof or top of the entry was not properly or sufficiently propped to prevent the stone or slate from falling, and that such defective condition of the roof had not been discovered or remedied owing to the negligence of the operator or of a person intrusted by it with the duty of seeing that the ways, works, machinery, or plant were in proper condition. An answer to such complaint filed by the operator is insufficient where it avers that the plaintiff was guilty of negligence which proximately contributed to his injury in that the roof of the entry was in a defective condition, the slate and other materials composing the roof of the entry being loose and liable to fall; that this defective condition of the roof existed at the place at which the miner was working at the time of his injury; that the miner knew of such defective condition of the roof and knew the danger of working beneath the same at the place at which he was working at the time of receiving the injury; but that notwithstanding this injury, the miner negligently worked in said entry beneath said defective roof, which negligence proximately contributed to his said injury, for the reason that the statute (section 3910) provides that in no event shall it be contributory negligence or assumption of risk on the part of the miner to remain in the employment of an operator after knowledge of the defect or negligence causing the injury, unless he be a miner whose duty it is to remedy the defect or who committed the negligent act causing the injury complained of.

Lookout Fuel Co. *v.* Phillips (Alabama), 66 Southern, 946, p. 947, November, 1914.
Burnwell Coal Co. *v.* Setzer (Alabama), 67 Southern, 604, p. 606, December, 1914.

APPLICATION OF COMMON-LAW RULES.

In an action by a miner for personal injuries because of the failure of the mine owner to furnish props as required by the statute of Missouri, a court can not apply the rules of contributory negligence or assumption of risk with the same strictness as in actions for common-law negligence as these would in effect nullify the statute; and unless, therefore, the danger was so apparent and imminent that the miner placing himself therein would amount to self-inflicted injury, he should not be denied recovery merely on a showing that there was some risk attending the further prosecution of the work and that he assumed such risk.

Runyan *v.* Marceline Coal & Mining Co. (Missouri Appeals), 172 Southwestern, 1165, p. 1167.

EFFECT OF AMENDMENT ON CONTRIBUTORY NEGLIGENCE AND ASSUMPTION OF RISK.

The only effect of the amendment of 1907 to subdivision 1 of section 3910 of the liability act of Alabama, was to remove as a basis of assumption of risk and of contributory negligence on the part of a miner in respect of a defective condition within the purview of such subdivision, the remaining in service after knowledge by the miner injured in consequence of the defect in the condition to which the complaint attributed the injury for proximate cause, of the defect in the condition of the ways, works, machinery, or plant of the operator, except in cases where the miner injured was under the duty to remedy the defect causing the injury, or where the miner injured committed the negligent act causing the injury complained of. There is no general legislative purpose expressed, or necessarily impliable, to deny the existence or the defensive effect of contributory negligence or assumption of risk in all cases.

Burnwell Coal Co. *v.* Setzer (Alabama), 67 Southern, 604, p. 606, December, 1914.

SUFFICIENCY OF ANSWER.

In an action for damages by an injured miner against a mine operator, under the statute of Alabama, an answer is insufficient if it fails to allege or show either that it was the duty of the injured miner to remedy the alleged defect causing the injury, or that the miner committed a negligent act which caused the injury complained of, or which contributed with the alleged defect in causing it, and more especially where the answer fails to show that the miner had knowledge of the alleged defect in time to avoid the injury.

Lookout Fuel Co. *v.* Phillips (Alabama), 66 Southern, 946, p. 947, November, 1914.

REQUEST FOR PROPS NOT EVIDENCE OF KNOWLEDGE OF DANGER.

Under the statute of Iowa, section 2489–5a, 16a, Code Supplement of 1913, it is made the duty of a mine operator to furnish props and timbers on request of the miner and it is made the duty of the miner to prop his working place and make the same safe; but the fact that a miner ordered props is not to be considered as evidence of knowledge of imminent or impending danger, as such a construction of the statute would mean that a miner could not work after ordering props until the props were actually received; but the miner is expected to foresee his needs to some extent and make his request so that the operator may have reasonable time to comply; and the statute implies that the roof of a mine may present latent and potential dangers at all times, though such dangers are not imminent or impending, and a miner is not to be charged with contributory negligence where after making a request for props he continues to work in his place, if by the inspection required by the statute his place is not so imminently dangerous as would deter a prudent man from working in such a place.

Edgren *v.* Scandia Coal Co. (Iowa), 151 Northwestern, 519, p. 522, March, 1915.

DUTY TO EXAMINE WORKING PLACE—DEFENSE.

In an action for the death of a miner caused by a portion of the roof of a mine falling upon him, an answer which avers that the intestate was guilty of negligence which proximately contributed to his death, in that he did not examine his working place under the rock or place that fell before commencing to work thereunder, which proximately caused his death; and averring that it was the duty of such intestate, before commencing work, to examine his working place, and his alleged death was the proximate result of his failure to perform such duty, is insufficient for failing to aver that an examination of the roof would have disclosed the defect, as well as the danger of going to work at the particular place, and for the additional reason that the averment of the failure of the deceased to examine the working place under the roof before commencing to work proximately caused his injury, is but a conclusion of the pleader.

Henderson *v.* Tennessee Coal, Iron & Railroad Co. (Alabama), 67 Southern, 414, p. 415, December, 1914.

VENTILATION OF MINE—FAILURE OF MINER TO OBEY INSTRUCTIONS.

A miner can not recover from a mine operator for injuries caused from the presence of poisonous gas in the mine, due to the negligence of the operator in failing to ventilate the mine, as required by the statute of Missouri, where the lack of proper ventilation was due to the failure of the operator to make an opening between two rooms of

the mine, where it appeared that the miner himself and his two sons were the only persons authorized to work in the particular room and where the miner had been directed by the mine operator to put in the cross cut in order to give the required ventilation of the mine.

Perry *v.* Northwestern Coal & Mining Co. (Missouri Appeals), 175 Southwestern, 140, p. 141, April, 1915.

EFFECT ON ASSUMPTION OF RISK.

OPERATOR'S BREACH OF STATUTORY DUTY—ASSUMPTION OF RISK.

Risks assumed by a miner are only such as arise after the mine operator has discharged his statutory duty.

Log Mountain Coal Co. *v.* Crunkleton (Kentucky), 169 Southwestern, 692, p. 694, October, 1914.

EFFECT ON ASSUMPTION OF RISK AND CONTRIBUTORY NEGLIGENCE.

The coal-mining statute of Colorado was drawn with the sole purpose to insure safety in the method and manner of mining and was intended to cure the rigors of the common-law rule as applied to assumption of risk and contributory negligence, and courts can not be less liberal in the interpretation of such statutes than the spirit of the law demands; and under the statute a miner is not to be charged with assumption of risk or be held guilty of contributory negligence in removing coal from one side of a section to another so as to leave no natural support across the (wall) face of the roof, nor through the fact that the chutes to one side of the section had caved in.

Lindquist *v.* Pacific Coast Coal Co. (Washington), 142 Pacific, 445, p. 448.

MINER CAN NOT WAIVE VIOLATION OF STATUTE.

In an action for damages for the death of a miner due to the alleged violation of the statute of Oklahoma on the part of the mine operator, the servant as a matter of law can not waive a compliance by the operator therewith and assume the risk of the operator's negligence in failing to comply with the statute, as to permit mine operators to avail themselves of such assumption of risk by their miners would be in effect to enable them to nullify a penal statute.

Great Western Coal & Coke Co. *v.* Coffman (Oklahoma), 143 Pacific, 30.
Great Western Coal & Coke Co. *v.* McMahan (Oklahoma), 143 Pacific, 23.
Great Western Coal & Coke Co. *v.* Cunningham (Oklahoma), 143 Pacific, 26.

VIOLATION OF STATUTE AS TO SAFETY APPLIANCES.

A miner injured while being carried down a shaft can not be charged with an assumption of risk where the mine operator violates a positive statute in not furnishing an iron-bonneted safety cage for lowering and hoisting miners.

Ryan *v.* Manhattan Big Four Mining Co. (Nevada), 145 Pacific, 907, p. 910, December, 1914.

ASSUMPTION OF RISK NOT A DEFENSE.

In an action for damages for the death of a miner the only question in issue under the statute of Oklahoma is whether or not the statute has been violated, and if so whether such violation was the proximate cause of the death, and it is immaterial whether the court gave conflicting or erroneous instructions upon the doctrine of assumption of risk, as the defense of assumption of risk is not available under the statute.

Great Western Coal & Coke Co. *v.* Cunningham (Oklahoma), 143 Pacific, 26, p. 28.
Great Western Coal & Coke Co. *v.* McMahan (Oklahoma), 143 Pacific, 23.
Great Western Coal & Coke Co. *v.* Coffman (Oklahoma), 143 Pacific, 30.

In an action for the death of a miner caused by a fall of rock from the roof of the chamber or station where he was working, the mine operator can not, under the employers' liability act of California, set up the defense of assumption of risk by the miner for the purpose of defeating a recovery.

Crabbe *v.* Mammoth Channel Gold Mining Co. (California), 143 Pacific, 714, p. 715.

FAILURE TO VENTILATE MINE—INJURY TO HEALTH.

In a mine insufficiently ventilated and in which dynamite and powder are used for shooting purposes, carbon dioxide or black damp, and carbon monoxide or white damp, may be generated. The former is commonly known as carbonic acid gas and is dead air with the properties of oxygen exhausted, is nonpoisonous, but causes suffocation by excluding oxygen from the lungs, while the white damp makes a person ache, dizzy, and sick at the stomach, and has the effect of destroying the red corpuscles of blood, the oxygen-carrying property of the blood; and if a person is deprived of a sufficient quantity of oxygen it results in a worn-down, debilitated, and weakened condition of the system; and where a miner who has been working in a mine is found to be in such condition because of the alleged negligence of the mine operator, in failing to ventilate the mine in accordance with the statute, the question of the operator's negligence in this respect is properly submitted to a jury for a determination and the plea of the assumption of risk on the part of the miner by continuing to work in the mine with a knowledge of the effect of the air upon his system, can not be interposed in an action based on the operator's failure to perform the statutory duty as to ventilation.

Log Mountain Coal Co. *v.* Crunkleton (Kentucky), 169 Southwestern, 692, p. 694, October, 1914.
Jellico Coal Mining Co. *v.* Walls (Kentucky), 170 Southwestern, 19, p. 20, November, 1914.
See Perry *v.* Northwestern Coal & Mining Co. (Missouri Appeals), 175 Southwestern, 140, April, 1915.

RISK OF OPERATOR'S VIOLATION OF STATUTE NOT ASSUMED.

A miner working under a contract whereby he was to receive 45 cents per ton for coal mined and loaded, was to be furnished all necessary props, caps, and headers, and was himself to set the props, caps, and headers as needed and take care of all slate, wherever he might find the slate in such condition as to make it unsafe to leave it, then he should take it down, and where slate examined by him seemed to be sound, and nothing to indicate it was about to fall, or would fall before props and headers requested and promised were supplied, it was not incumbent upon the miner to take it down, and it can not be said that either under his contract or the statute he assumed the risk of its falling before the props and headers were furnished.

Big Branch Coal Co. *v.* Wrenchie (Kentucky), 170 Southwestern, 14, p. 16, November, 1914.

DUTY TO PROP ROOF OF ENTRY.

The statutes of Alabama on the subject of mining indicate a public policy to the effect that mine owners and operators shall be charged with the duty of making their mines reasonably safe for miners; and these statutes require miners themselves in certain cases to look out for their own safety, as in propping the roofs of rooms in which they work, but impose on the operator the duty of furnishing the timbers for such propping; but the statutes do not require miners to prop or look after the safety of entries, and that duty therefore rests on the owners or operators of the mines. Section 1021 of the code requires owners and operators of mines to keep a sufficient supply of props or other timbers used in the mine so that the miners may at all times be able to prop their working places, and the mine owner or operator shall afford miners the proper facilities for the delivery of props and other timbers needed by them in their respective working places. But the duty to prop an entry may rest on the miner himself, when he is employed to do that particular work or is employed to make the entry itself and to prop it as the work progresses, or where he engages in work in the entry, after it is made and properly propped, which is of such a character as to render further propping necessary; and in all such cases the miner assumes the risk of his negligent failure to do the necessary propping, and if injured as the result of such failure he can not recover from the operator.

Lookout Fuel Co. *v.* Phillips (Alabama), 66 Southern, 946, p. 948, November, 1914.

REQUEST FOR PROPS—FAILURE TO FURNISH—CONTINUING WORK.

Section 4999-3a, Code Supplement of 1913, of Iowa, provides that where the property, works, machinery, or appliance of an employer are defective or out of repair, and where it is the duty of an employer

to furnish reasonably safe machinery, appliances, and place to work, an employee shall not be deemed to have assumed the risk by continuing in the prosecution of the work growing out of any such defects, unless the danger is imminent and to such an extent that a reasonably prudent person would not continue in the prosecution of the work and it is the purpose of this statute to prevent an employer or a mine operator from laying upon his employee or miner the consequences of his own failure simply because such failure was known to the employee or miner, and under this statute a miner will not be held to have assumed the risk though he continues working in a mine partially unpropped, where he had made the statutory request on the operator for props and the operator had failed to comply with such request and the miner was injured because of such failure of the operator to furnish the props requested.

Edgren *v.* Scandia Coal Co. (Iowa) 151 Northwestern, 521, p. 523, March, 1915.

NEGLIGENCE OF MINE SUPERINTENDENT.

PROOF OF NEGLIGENCE OF SUPERINTENDENT—PROXIMATE CAUSE.

In an action against a mine operator for damages for negligence because of the negligence of the mine operator's superintendent, and in the absence of a charge of willfulness, wantonness, or intentional injury, it was not error for a court to instruct a jury to the effect that the superintendent's directions to the miner to pinch or prize the coal in the wall, and to turn the water on, are alone sufficient to justify a recovery; but the jury must further find that in giving such instructions the superintendent was at fault and that such fault was the proximate cause of the miner's injury; and that the statute of Alabama relating to liability of employers for damages when their employees are injured while engaged in the work for which they are employed, does not authorize the jury to award damages unless the operator's agent was guilty of negligence.

Barker *v.* Tennessee Coal, Iron & Railroad Co. (Alabama), 66 Southern, 600, November, 1914.

DELEGATION OF DUTY.

DUTY OF FOREMAN TO INSPECT—DELEGATION OF DUTY—NEGLIGENCE.

The statute of Oklahoma imposes upon a mine foreman or his assistant the duty of daily inspection, and he has no right to delegate this duty to the fire boss or to any other person working in the mine, other than a regular assistant; and this duty is a nondelegable one, and the failure to discharge which constitutes primary negligence on the part of the mine operator.

Rock Island Coal Mining Co. *v.* Davis (Oklahoma), 144 Pacific, 600, p. 603, December, 1914.

SAFE PLACE—DELEGATION OF DUTY.

The owner and operator of a mine may delegate to an electrician the duty or authority to make safe or properly protect electric wires through a passageway in his mine through which the miners were required to pass; but the mine operator can not, under his duty to furnish a safe working place, delegate such duty in such a manner as to relieve himself of liability for failure to furnish a safe working place.

Hazzard *v.* Consolidated Coal Co. (Michigan), 149 Northwestern, 991, p. 992, December, 1914.

DELEGATION OF VENTILATION OF MINE.

The statute of Missouri imposes upon a mine operator the duty of properly ventilating the mine, and a breach of such duty is negligence per se, and the operator can not delegate the performance of this duty to another and thereby shift responsibility or liability for its nonperformance; but the statute does not place upon the operator of a mine the duty of an insurer, but does impose upon him the positive duty to exercise reasonable care to provide a sufficient amount of ventilation. The rule that the responsibility of the mine operator can not be shifted does not imply that the operator must perform the task of making a safe place or providing for the ventilation for the miner with his own hands, or that he is forbidden to require a miner to keep his own place in order, though the task be within the scope of the miner's skill and experience. It is not an attempted delegation of duty, but is in fact a step in the performance thereof when a mine operator, in the exercise of his legal right to direct a miner to do any and all things within the scope of the general employment, directs him to do that which, when done, will make safe the place of the miner's employment. The fact of directing a miner to dig coal from a partition wall, to make an opening therein for the purposes of ventilation, is not different from digging coal in any other part of a room, and the operator in ordering a miner to do such work for the purpose of securing proper ventilation, was not delegating a duty, but was requiring a service the operator had a right to demand of the miner.

Perry *v.* Northwestern Coal & Mining Co. (Missouri Appeals), 175 Southwestern, 140, p. 142, April, 1915.

NONDELEGABLE DUTIES—NEGLIGENCE.

The duty of a mine operator to exercise reasonable care and diligence to provide his miners with a reasonably safe place in which to work, having regard to the kind of work and the conditions under which it must necessarily be performed, is an absolute one, and if instead of performing this duty in person, the operator delegates it to an officer or servant, then such officer or servant stands in the

place of the operator, and the negligence of such officer or servant is the negligence of the operator himself, and any miner injured by such negligence may recover, regardless of the relation the injured miner sustained to the officer or servant whose negligence resulted in inflicting the injury.

Rock Island Coal Mining Co. *v.* Davis (Oklahoma), 144 Pacific, 600, p. 605, December, 1914.
Hazzard *v.* Consolidated Coal Co. (Michigan), 149 Northwestern, 991, December, 1914.

STATUTORY ACTION FOR WRONGFUL DEATH.

ACTION FOR DEATH OF MINER—STATUTE OF LIMITATIONS.

An action for the wrongful death of a miner must under the statute of Arkansas be brought within two years for the death of such miner, and the time for commencing such action is not extended by the statute authorizing a second suit within one year where a plaintiff has been nonsuited in his first action.

Western Coal & Mining Co. *v.* Hise, 216 Fed., 338.

PERSONS ENTITLED TO SUE.

The State of New York gives a right of action for wrongful death and provides for the maintenance thereof by the personal representatives of the deceased for the benefit of those entitled to recover, and an action for the death of a husband and father wrongfully killed in Pennsylvania is properly brought and maintained in New York by the administrator of the decedent; but if there is no widow or child and the father and mother survive who are entitled to the benefit of the recovery, then an action must be brought by them, and not by an administrator.

Teti *v.* Consolidated Coal Co., 217 Fed., 240, p. 450.

RIGHT OF ALIEN TO SUE.

The treaty of July 3, 1913, between the United States and Italy, giving a citizen of Italy the right to sue for wrongful death, gives an alien coming into New York and prosecuting his action for wrongful death precisely the same rights and remedies, and no more, which the State of New York accords to its own citizens.

Teti *v.* Consolidated Coal Co., 217 Fed., 443, p. 450.

PLACE OF BRINGING SUIT.

Whether an action for the wrongful death of a miner against a Maryland mining corporation operating a mine in Pennsylvania and authorized to do business in the State of New York should be brought and maintained in the courts of the latter State for a death occurring in Pennsylvania must be ruled by the procedure of New York, as the

general rule is that the procedure and remedy are regulated by the law of the forum.

Teti *v.* Consolidated Coal Co., 217 Fed., 443, p. 447.

An action against a Maryland corporation operating coal mines in Pennsylvania and authorized to transact business in New York may be maintained in either Pennsylvania, Maryland, or New York, under the statute authorizing transitory actions to be brought wherever a defendant is found.

Teti *v.* Consolidated Coal Co., 217 Fed., 443, p. 447.
See Lehigh Valley Coal Co. *v.* Yensavage, 218 Fed., 547.

LINEAL HEIRS—FATHER AND MOTHER.

The statute of Colorado does not exclude the father and mother from the term "lineal heirs" as a lineal heir is one who inherits in a line either ascending or descending from the common source as distinguished from a collateral heir.

Rocky Mountain Fuel Co. *v.* Kovaics (Colorado), 144 Pacific, 863, p. 865, December, 1914.

RIGHT OF FATHER OR MOTHER TO SUE.

Section 649 of the Revised Statutes of 1908 (Colorado), known as the coal-mining act, provides that a right of action shall accrue to the widow and lineal heirs of a person whose life shall be lost, and in an action for the death of a miner this statute can not be held to mean child or children only, but when construed in connection with the Colorado statutes of descent, includes the father and mother, as under the statutes of descent the father and mother inherit from a child and are specifically authorized to sue for the death of a child, and by the insertion of the word "lineal" before the word "heirs" in the coal-mining act, the legislature intended to exclude collateral heirs only, and to limit the right of action to those who were entitled to receive the benefits of the person killed, while he lived, and to enable them to recover damages for the loss of such benefits in the case of his death, and to exclude the father and mother would deprive them of a right under the coal-mining act, to which they are entitled under other statutes of Colorado.

Rocky Mountain Fuel Co. *v.* Kovaics (Colorado), 144 Pacific, 863, p. 865, December, 1914.

VIOLATION OF STATUTORY DUTY—STATUTORY REMEDY.

An action for the death of a miner caused by the alleged breach of the statutory duty on the part of the coal operator to furnish props, must be brought under the coal-mining act, as this is the only act that specifically gives the right of action for negligence in failing to furnish props.

Rocky Mountain Fuel Co. *v.* Kovaics (Colorado), 144 Pacific, 863, p. 866, December, 1914.

INJURY FROM FAILURE TO PROVIDE SAFE WORKING PLACE.

A complaint in an action by an injured miner under the statute of Alabama is sufficient where, in the usual terms, following the statute, it ascribes the injury to the defect in the ways, works, machinery, or plant of the operator, and avers that the defect consisted in this, that a wall of ore and clay, near which the miner was engaged in the performance of his duty at the time he received the injury, was unsafe and insecure, so that a large embankment therefrom fell upon the miner and caused the injury complained of.

Sloss-Sheffield Steel & Iron Co. *v.* Terry (Alabama), 67 Southern, 678, p. 680, December, 1914.

ACTION BY PARENT FOR DEATH OF CHILD—PLEADING AND PROOF.

The statute of Missouri gives the parents the right to sue for the wrongful death of a child; and in an action by a father and mother for the death of a child they recover in tort on the right which the child would have had if he had survived the injury, and which right died with the injured child at common law but has been by the statute expressly transmitted to the parents; and in such case it is not necessary to aver or prove the amount of damages, but the jury must make their estimate of damages from the facts proved, and they exercise their own judgment upon such facts by connecting them with their own knowledge and experience which they are supposed to possess in common with the generality of mankind.

Dalton *v.* St. Louis Smelting & Refining Co. (Missouri Appeals), 174 Southwestern, 468, p. 470, March, 1915.

PROOF OF INDIGENT CONDITION OF BENEFICIARY.

The statute of California, section 1970, Civil Code, gives the right of action to a personal representative of a person killed by the wrongful act of another and provides for the recovery of damages for the benefit of the widow, children, dependent parents, and dependent brothers and sisters; and in an action for the wrongful death of a minor against a mine operator for the benefit of the father of the deceased, evidence of the financial condition of the father is admissible, and the word "dependent" as used in the statute was intended to describe a condition of actual dependency, and not a dependency that rested on a presumption on account of relationship, and it was necessary for the plaintiff to prove, not a mere relation of dependency, but actual dependency.

Balakalala Consolidated Copper Co. *v.* Reardon, 220 Fed., 584, p. 588, February, 1915.

Paskvan *v.* Allouez Mining Co. (Michigan), 152 Northwestern, 82, p. 84, April, 1915.

SUFFICIENCY OF ANSWER.

Where a complaint, in an action for the death of a licensee while in the mine of the defendant operator and at its invitation and request, alleges that the licensee was killed by a portion of the roof of the mine falling upon him, and that his death was proximately caused by the negligence of the operator's servants while acting within the line and scope of their employment, and that they negligently caused a portion of the said roof to fall upon the intestate, an answer which avers that the intestate knew of the defect or negligence which caused his death, and failed within a reasonable time, to give information thereof to the operator or to some person superior to such intestate engaged in the service or employment of the defendant, is insufficient for failure to aver that the intestate was charged with the duty of informing the operator of the alleged dangerous condition of the roof.

Henderson *v.* Tennessee Coal, Iron & Railroad Co. (Alabama), 67 Southern, 414, p. 415, December, 1914.

MINES AND MINING OPERATIONS.

NEGLIGENCE OF OPERATOR.

DEFINITION OF NEGLIGENCE.

Negligence in a legal sense is a failure on the part of a mine operator to observe for the protection of the interests of the miner that degree of care, precaution, and vigilance which the circumstances justly demand, whereby the miner suffers injury.

Darby Coal Mining Co. *v.* Shoop (Virginia), 83 Southeastern, 412, p. 416, November, 1914.

NEGLIGENCE OF OPERATOR AND THIRD PERSON—EFFECT.

Where an injury is sustained by a miner as the result of the operator's negligence in failing to provide him with a reasonably safe place to work, or reasonably safe machinery or tools with which to perform his work, although such negligence would not have caused the injury complained of, but for the intervening negligent act of another miner in the same field of service with the injured miner, the fellow servant doctrine can not be invoked by the operator to escape liability for the injury, as in such case the negligence of the fellow servant will be treated as combining with that of the master in causing the injury, or he will be regarded as the vice principal of the operator and his negligence imputed to the operator, and in either case the primary negligence of the operator will be regarded as the proximate cause of the injury.

Fluhart Collieries Co. *v.* Meets (Kentucky), 169 Southwestern, 686, p. 687, October, 1914.

DEGREE OF CARE REQUIRED—LIABILITY.

A person guilty of negligence or an omission of duty should be held responsible for all the consequences which a prudent and experienced man fully acquainted with all the circumstances which in fact exist, whether they could have been ascertained by reasonable diligence or not, or would have thought at the time of the negligent act as reasonably possible to follow, if they had been suggested to his mind.

Producers' Oil Co. *v.* Eaton (Oklahoma), 143 Pacific, 9, p. 10.

DEGREE OF CARE—DANGERS NOT ANTICIPATED.

The operator of a mine is not to be charged with negligence because he worked it on the pillar system, supporting the roof with props, where a sufficient supply of timber was furnished at the entrance of the working place; and an operator can not be held liable for the death of a miner resulting by the fall of a portion of the roof due to a "squeeze" because he failed to provide for cribbing instead of props, and especially where the mine was not one that is called an unsafe mine; and the fact that the proof showed that the accident would not have happened if cribbing had been used is not of controlling influence, as such testimony is given in the light of what has occurred and is of little value in determing whether, when considering the character of the working place and the past history of the mine, the "squeeze" should have been anticipated and guarded against by cribbing instead of props.

Lindquist *v.* Pacific Coast Coal Co. (Washington), 142 Pacific, 445, p. 447.

PROXIMATE CAUSE OF INJURY.

A charge of negligence in an action by an injured miner against a mine operator in failing to furnish a linchpin, and by reason of which the wheel of a coal car came off and the miner was injured while reaching for the wheel inside of the shaft, is not supported by proof that the miner while reaching inside of the shaft for the car wheel which had come off of the car because there was no linchpin and was while projecting his head and body inside of the shaft struck on the head by falling material, the cause of the falling material being unknown, there being no causal connection between the alleged negligence on the part of the operator in failing to furnish a suitable linchpin and the falling material that actually produced the injury. The rule is that an injury for which damages are sought must be the natural and probable consequence of the negligence charged and must be of such a character as an ordinarily prudent person ought to have foreseen, and might probably occur as a result of the alleged negligence.

Jenkins *v.* La Salle County Carbon Coal Co. (Illinois), 106 Northeastern, 186, p. 187, October, 1914.

A coal-mining corporation is not liable for the death of a miner who, in violation of a rule of the mine, attempted to pass through a cage at the bottom of the shaft and was killed by a sudden starting of the cage, and where the rule required the miners at the bottom of the shaft to use the passageway around the cage in passing from one side of the shaft to the other; and negligence can not be imputed to the mining company because of its failure to equip its mine and shaft with a speaking tube or telephone connection, where the want of such speaking tube or telephone connection in no manner contributed to the death of the miner.

Colorado Capitol Coal Mining Co. *v.* Chatfield (Colorado), 143 Pacific, 1095, p. 1096.

PROXIMATE CAUSE OF INJURY—QUESTION OF FACT.

The question as to whether or not a negligent act involving a breach of duty on the part of an oil-drilling company to its employee is the proximate cause of an injury to such employee is one of fact for the determination of a jury, if there is any evidence whatever reasonably tending to prove the same.

Producers' Oil Co. *v.* Eaton (Oklahoma), 143 Pacific, 9, p. 11.

ACCIDENT AS PROXIMATE CAUSE OF INJURY.

A coal-mining company loading coal from its tipple into a car standing on a spur or side track is not liable for an injury to a person walking along the main track opposite the tipple at a distance of 18 feet from the side track on which the coal car was standing, where the injury was caused by a single block or lump of coal bounding over the side of the car and striking the traveler on the leg, though an employee of the coal company had stated to the traveler that it was safe to walk on the main track past the tipple and the coal car in process of loading, as in the particular case the proximate cause of the injury was a pure accident and not the result of actionable negligence on the part of the coal company.

Kirk *v.* West Virginia Colliery Co., 215 Fed., 77, p. 78.

QUESTIONS OF FACT.

The questions of the negligence of a coal company and the contributory negligence of a miner are questions to be determined by the jury where the evidence shows that the miner lost his life while engaged with other employees in making repairs upon the coal tipple, and while standing upon a platform used for making the repairs and attempting to remove an iron shaft of the tipple in order that it might be replaced by a larger shaft a plank of the platform upon which the miner was standing slipped from a girder upon which it rested, causing him to be thrown down to a coal car some

25 feet below the platform, causing his death, and where it appeared that the repairs were being made in the night and the only light was that furnished by miners' lamps in the caps of the workmen.

Interstate Coal Co. *v.* Shelton (Kentucky), 169 Southwestern, 546, October, 1914.
Darby Coal Mining Co. *v.* Shoop (Virginia), 83 Southeastern, 412, p. 415, November, 1914.

Where according to the custom of a mine it was the duty of the operator to prop the roof of a roadway, the questions of the operator's negligence in this respect and of the miner's contributory negligence are for the jury to determine and can not be passed upon by a court as a matter of law.

Stringer *v.* York (Kentucky), 170 Southwestern, 527, November, 1914.

RELATION OF MASTER AND SERVANT—MINER RIDING OUT OF MINE AFTER WORK.

A coal-mine operator can not escape liability for the death of a miner caused by a shot and a subsequent explosion of coal dust while the miner with others was being hauled out of the slope after he had ceased his work in the mine on the theory that the relation of master and servant terminated at the time the miner quit work in the mine and the relation did not exist while he was being hauled out of the slope of the mine and no duty was owing him by the operator; but as a matter of law the relation of master and servant under such circumstances continued until the miner was taken from the mine.

Great Western Coal & Coke Co. *v.* McMahan (Oklahoma), 143 Pacific, 23.
Great Western Coal & Coke Co. *v.* Cunningham (Oklahoma), 143 Pacific, 26.
Great Western Coal & Coke Co. *v.* Coffman (Oklahoma), 143 Pacific, 30.
Great Western Coal & Coke Co. *v.* Boyd (Oklahoma), 143 Pacific, 36.
Great Western Coal & Coke Co. *v.* Belcher (Oklahoma), 143 Pacific, 36.
San Bois Coal Co. *v.* Resetz (Oklahoma), 143 Pacific, 46.

FAILURE TO INSPECT ROOF.

It is a matter of common knowledge where shots are fired in a mine and where the roof is of such height that miners can not ascertain for themselves whether they are working in an obviously dangerous place proper inspection should be made and trimming or brushing done from time to time as needed to make the place reasonably safe, and in case of injury to a miner by a fall of slate or rock it is fairly inferable that the negligence of the operator in failing to inspect and trim or brush the roof at reasonable intervals was the proximate cause of the injury, and the jury may fairly infer that an ordinary inspection of the roof would have revealed the loose condition of the rock in time to have prevented the injury.

Scott *v.* American Lead & Smelting Co. (Missouri Appeals), 173 Southwestern, 23, p. 26, January, 1915.

INJURIES FROM BLASTING.

A mining company in blasting rock preparing for the construction of a tipple for its mining plant is liable for damages to a child caused from the rock thrown by the blast, under the rule that where blasting operations result in a direct trespass upon the premises injured by casting soil or rock thereon, the liability of the person causing the injury is absolute, and it must respond in damages irrespective of the question of negligence or want of skill, and without regard to whether precautions are used or not to prevent the injury complained of, as the act itself is a nuisance, and this liability attached to the mining company although the house in which the parent lived and in which the child was injured belonged to the mining company.

Allegheny Coke Co. *v.* Massey (Kentucky), 174 Southwestern, 499, March, 1915.

NO LIABILITY FOR ACCIDENTAL INJURIES.

A mine operator is not liable for an injury to a miner who was sent with another miner to load a barrel of oil on a car and where the miner while in the act of jumping from the car to avoid danger was injured by a heavy plank sliding down a hillside, which had been accidentally dropped by two other miners coming to assist in loading the barrel of oil on the car and who were carrying the plank for the purpose of facilitating the loading of the barrel of oil.

Farris *v.* Cabin Creek Consolidated Coal Co., 220 Fed., 813, p. 815, February, 1915.

FAILURE TO INSPECT FOR MISSED SHOTS.

The rule that a mine operator is not required to furnish a safe place for miners, where the perils to the working place are caused by the progress of the work in which the miners are engaged, does not apply to persons engaged in drilling holes for firing shots, as such work is not work of construction or repair and in which the risks are caused by the progress of the work and are assumed by the miner; but the operator directs the drillers to work in places prepared for them as they are moved about the mine from drift to drift, and the operator has undertaken to inspect each working place before assigning the drillers to the work there; and under such circumstances an operator is liable for failure to give proper inspection to discover "missed shots," where such missed shots could have been discovered by inspection, and the operator is not relieved from the duty of inspection and of liability for failure to inspect for shots simply because drillers themselves do generally look out for missed shots.

Balakalala Consolidated Copper Co. *v.* Reardon, 220 Fed., 584, p. 589, February, 1915.

ACTIONS—PLEADING AND PROOF OF NEGLIGENCE.

EFFECT AND WEIGHT OF EXPERT EVIDENCE.

In an action by an injured employee for damages caused by the alleged negligence of an oil drilling company, it is error for the court to instruct the jury that no reliance is to be placed on or aid to be gained from the opinion evidence of experts; but the jury may be instructed that they may disregard such evidence if they deem it unreasonable or not entitled to belief because of other and contradicting evidence from witnesses claiming positive knowledge.

Producers' Oil Co. *v.* Eaton (Oklahoma), 143 Pacific, 9, p. 11.

PLEADING NEGLIGENCE—VARIANCE.

In an action by an injured miner for damages there can be no recovery under an allegation of the failure of the mine operator itself to prop the roof where the miner was working and of its failure to furnish the miner a reasonably safe place to work, where the evidence shows a failure on the part of the operator to furnish the miner sufficient props and caps to support the roof; and under such allegations evidence of the failure of the operator to furnish the miner props and caps to support the roof is not admissible.

Palmer *v.* Empire Coal Co. (Kentucky), 172 Southwestern, 97, p. 98, January, 1914.

INJURY FROM COAL-DUST EXPLOSION—SUFFICIENCY OF COMPLAINT.

A complaint in an action by a miner injured by a coal-dust explosion is sufficient where it sets out the several duties cast upon the mine operator by the statute of Illinois, but alleges one breach only, and that the operator willfully failed to see that a certain roadway which was so dry that the air therein was clouded with dust was thoroughly sprinkled, sprayed, or cleaned as the statute requires, whereby the explosion and consequent injury to the plaintiff followed.

Eldorado Coal & Mining Co. *v.* Mariotti, 215 Fed., 51, p. 55.

SUFFICIENCY OF COMPLAINT BY INJURED MINER.

A complaint in an action by an injured miner against a mine operator is sufficient where it charges a defect in a chain, and that it was weak and not sufficient for the purpose for which it was furnished and averring that it was too weak, and insufficient, to perform the functions for which it was furnished and that it broke while being used and by reason of which the injury occurred.

Southern Iron & Steel Co. *v.* Boston (Alabama), 66 Southern, 684, p. 685, November, 1914.

INSUFFICIENCY OF COMPLAINT BY INJURED MINER.

A complaint in an action for personal injuries for the alleged negligence of a mine operator for failure to exercise reasonable care in furnishing a miner a reasonably safe place to work, is insufficient where it does not charge any inherent defect or knowledge in the plant, ways, or works, or any other neglect by the operator in the performance or discharge of a nondelagable duty, and where the negligence ascribed was the failure to discharge, by some servant, the delegable duty of properly or sufficiently fastening a cable to the trip or chain, and where there was no charge that the cable was not adaptable to the use for which it was furnished, or that it was defective or dangerous, as from all the allegations, the works may have been perfectly safe and the cable sound and sufficient and the only negligence was the failure of a fellow servant to properly fasten the cable to the trip or chain so that it would not come loose.

Southern Iron & Steel Co. *v.* Boston (Alabama), 66 Southern, 684, p. 685, November, 1914.

INJURY TO LICENSEE—WILLFUL INJURY.

In an action by a licensee for an injury while riding on a tramcar in a mine, a complaint is insufficient which alleges that the engineer or servant of the mine operator, who was in charge and control of the engine and tramcar, wantonly, willfully, or intentionally wrecked and derailed the tramcar upon which the plaintiff was riding in the mine, and that by reason of such wanton, willful, or intentional act of the engineer the plaintiff was badly injured, but which fails to aver that the engineer knew that the plaintiff was on the car, as the averments only characterizes the act and not the injury as wanton or willful.

Woodward Iron Co. *v.* Finley (Alabama), 66 Southern, 587, November, 1914.

ALLEGATIONS AS TO SAFE PLACE.

In an action by a miner against a mine operator for damages for injuries to the miner on the ground of the alleged negligence of the mine operator, it is sufficient where it avers that the alleged negligence related to employment in operating a coal mine and the negligence of the operator to use reasonable care to provide a reasonably safe place to work; and if act of the negligence resulting in the injury was that of a fellow servant, this is a matter of defense, not requiring a negative averment in the complaint. It is also unnecessary to aver ignorance on the part of the plaintiff of the dangerous character of the appliance or want of means of ascertaining its dangerous character, as the miner in such case has the right to assume that the operator has performed his duty unless the danger be so open and apparent that the miner is bound to see and thereby assume the risk; but even in

such case no negative averment is required, as this is a matter of defense.

Yates v. Crozer Coal & Coke Co. (West Virginia), 84 Southeastern, 626, p. 627, March, 1915.

INJURY TO MINER—EXCLUSIVE REMEDY.

An injured miner can not in the same complaint, by different counts recover for compensation for his injury under the workman's compensation law and under another count recover damages for pain and suffering and for disfigurement of his body from the same injury and resulting from the negligence of a mine operator, as the remedy under the compensation act is exclusive.

McRoberts v. National Zinc Co. (Kansas), 144 Pac. 247, November, 1914.

PLEADING AFFIRMATIVE DEFENSE.

Contributory negligence in the State of California is an affirmative defense which must be pleaded and while a pleader may charge contributory negligence in general terms, yet he is requiréd to plead the facts of such negligence and the facts pleaded must show a causal connection with the injury. Contributory negligence in law is predicated on the existence of negligence on the part of the defendant, the responsibility for which he seeks to avoid by showing a concurrent and contributing negligence upon the part of the person injured.

Crabbe v. Mammoth Channel Gold Mining Co. (California), 143 Pacific, 714, p. 716.

RECOVERY MUST BE BASED ON ALLEGATIONS OF THE PLEADING.

A miner suing for injuries caused by the presence of poisonous gas in his working room in the mine, caused by the alleged breach by the mine operator of the statutory duty "to provide and maintain a good and sufficient amount of ventilation," in that the mine operator negligently failed to put an opening in the partition wall between two rooms, by means of which proper ventilation might have been supplied, can not recover where it appears from the evidence that the lack of the opening referred to in the pleading did not contribute in any way whatever to the plaintiff's injuries for the reason that there was no air circulating in the entry to which the opening would have led, but the lack of ventilation was due to the fact that there had been "a short circuit" in the main air current, produced by the enlargement of a ventilating door or hole in a different part of the mine and which prevented the air from circulating in the part of the mine where the miner was engaged.

Perry v. Northwestern Coal & Mining Co. (Missouri Appeals), 175 Southwestern, 140, p. 143, April, 1915.

PROOF OF CUSTOM INADMISSIBLE.

Under a contract between a mine owner and a miner, by which the miner was given the exclusive right to mine all the coal in a certain defined territory but stating no time within which the coal should be mined, testimony is inadmissible for the purpose of showing a general custom or usage in the particular mining district to the effect that contracts of miners with operating companies for the mining and removing of coal were subject to termination at the will of either party, as evidence of a custom or usage is not admissible if it contradicts expressly or by implication the terms of a contract.

Pratt Consolidated Coal Co. *v.* Short (Alabama), 68 Southern, 63, p. 67, February, 1915.

REPORT OF MINE INSPECTOR AS EVIDENCE.

In an action against a mine operator for damages for the death of a motor driver or engineer, due to an alleged defect in the motor track, the district mine inspector's report is not proper evidence on the trial of the case, particularly so where the mine inspector is present in person and testified as to what he knew about the condition of the mine.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 944, April, 1915.

EVIDENCE TO CONTRADICT A MINE FOREMAN.

In an action against a mine operator for the death of a trip driver caused by an alleged defect in the motor track, causing the motor to be derailed, knocking down a prop, and causing loose rock to fall upon and kill the driver, it is proper to show by a witness, for the purpose of contradicting evidence of the mine foreman, that the mine foreman had told the witness shortly after the accident that it was caused by a bad joint in the track.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 943, April, 1915.

DUTY TO FURNISH SAFE PLACE.

DILIGENCE COMMENSURATE WITH DANGER.

It is the duty of a mine owner to exercise ordinary care to provide a reasonably safe place in which the miners may perform their work, and he must use diligence to keep the place in a reasonably safe condition, and the diligence required must be commensurate with the character of the services required and with the dangers a prudent man would comprehend under the circumstances of each particular case, and a mine owner or operator is liable for any injury resulting from a failure to exercise such care; and it is the corresponding duty of a miner employed to exercise that degree of care

which is commensurate with the character of his occupation and that which a reasonably prudent person would use under like circumstances to protect himself from injury.

Rock Island Coal Mining Co. *v.* Davis (Oklahoma), 144 Pacific, 600, p. 603, December, 1914.

DUTY TO FURNISH SAFE WORKING PLACE—NONDELEGABLE DUTY.

Tunnels and entries in a mine are parts of the plant that the mine owner or operator must furnish to his miners and he must make them of sufficient width and height to permit cars to enter and leave the mine and provide sufficient space between the tracks and ribs or walls of coal to protect miners at their work; and the duty to provide proper passageways in a tunnel is a nondelegable duty imposed upon the mine owner or operator, and it is not one of the statutory duties imposed upon the mine foreman. The duty of providing reasonably safe haulageways with sufficient space to allow a driver to pass between the cars and at least one wall of the mine remains in the mine owner or operator, and his liability depends upon whether he complies with the requirements of the law, and in the absence of legislative demands he must make such haulageways reasonably safe for the purpose for which they are used.

Watson *v.* Monongahela River Consolidated Coal & Coke Co. (Pennsylvania), 93 Atlantic, 625, p. 626, January, 1915.

FAILURE TO PROVIDE REFUGE HOLES.

The operator of a mine may be liable on a charge of negligence for the death of a coal miner struck while on a slope in a coal mine by a running car on the ground of negligence in the failure to provide places of refuge or other safety devices in the slope where the deceased miner and others were required to pass in and out of the slope at the time cars were being operated upon a track in the slope.

Parkdale Fuel Co. *v.* Taylor (Colorado), 144 Pacific, 1138.

DUTY OF OPERATOR TO PROVIDE REFUGE HOLES.

The statute of Indiana, section 8581, Burns's Annotated Statutes, 1914, requires mine operators to provide places or refuge holes in the side walls of the mine of every haulageway used as a passageway by employees in traveling to and from their work where there is not a clear space of at least 3 feet in width between the side of the car and the wall or rib; and a complaint in an action by a miner for an injury is sufficient where it avers that the operator failed to provide such places of refuge and the space was less than 3 feet in width and the miner was injured while passing along the haulageway; and the complaint need not allege that it was practicable to construct

places of refuge without interfering with the working of the mine; nor is it necessary to allege that the operator had actual or constructive notice that the space between the side wall and the car was less than 3 feet, as the violation of a statutory duty is negligence per se.

Vandalia Coal Co. *v.* Coakley (Indiana Appeals), 180 Northeastern, 382, p. 384, March, 1915.

FAILURE TO MARK DANGEROUS PLACE—PROXIMATE CAUSE.

The fact that miners are engaged, under the direction of the mine manager, in cleaning up a coal mine after a shutdown of several months, and working under his general direction to make dangerous places safe does not relieve the mine operator of the statutory duty of having the mine examined by the proper mine examiner and of marking the dangerous places. The question of whether the failure of the mine examiner to mark the dangerous places was the proximate cause of the injury is a question of fact for a jury to determine on the trial of the cause.

Wilson *v.* Danville Collieries Co. (Illinois), 106 Northeastern, 194, p. 196, October, 1914.

INJURY FROM FALL OF ROOF—PRIMA FACIE CASE.

Proof that a coal miner was injured by a fall of slate from the roof of an entry which it was the operator's duty to maintain and keep in a reasonably safe condition, and that the miner had not removed any coal from the particular part of the entry and had done no act that could have occasioned the fall of the slate, is sufficient to make a prima facie case of negligence under the doctrine of res ipsa loquitur.

Main Jellico Mountain Coal Co. *v.* Young (Kentucky), 169 Southwestern, 841, p. 842, October, 1914.

COMPLETED CHAMBER.

The rule that a mine operator is not required to furnish a reasonably safe working place to a miner where the conditions of the working place are constantly changing does not apply to a completed chamber or station, and where there are no such changing conditions in the chamber or station which make it impracticable to make it safe, and especially where the mine operator had timber lying unused at the mouth of the shaft that would unquestionably have made the place safe.

Crabbe *v.* Mammoth Channel Gold Mining Co. (California), 143 Pacific, 714, p. 716.

SAFE PASSAGEWAY.

The relation of mine owner or operator and a miner raises a legal duty upon the part of the operator to furnish the miner with a reasonably safe place in which to work, and this duty carries with it the

obligation to see that a passageway through the mine to and from the miner's working place was reasonably safe.

Hazzard *v.* Consolidated Coal Co. (Michigan), 149 Northwestern, 991, p. 992, December, 1914.

DUTY OF OPERATOR TO MAINTAIN MOTOR TRACKS IN MINE.

Motor tracks in a main haulageway of a coal mine partake of the nature of both a place and an appliance, and as they are permanently laid in the mine they therefore become a part of the place and are essential to the operation of the motor and therefore an appliance for the removal of coal, and it is the duty of the mine operator to maintain his motor and motor tracks in a reasonably safe and suitable condition for the safety of an employee operating a motor in hauling coal out of a mine.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 942, April, 1915.
Crockett *v.* Keystone Coal & Coke Co. (West Virginia), 84 Southeastern, 948, April, 1915.

RELATIVE DUTY OF OPERATOR AND FOREMAN AS TO MOTOR TRACKS.

The statute of Alabama requires a mine operator to employ a mine foreman and prescribes the duties of the mine foreman to be, among other things, to keep and carefully watch over the ventilating apparatus and the airways, travelingways, pumps, and drainage, and to see that proper break-throughs are made, brattices used, no loose coal, slate, or rock is hanging overhead or along the haulageways, and sufficient props, caps, and timbers are furnished to miners, the water drained out of the working places and recesses made along the haulageways of not less than 100 feet apart between the wagon and the ribs for refuge, and provide a proper system of signals and lights where hauling is done by machinery of any kind; but nowhere is the foreman expressly given supervision of the motor tracks, and he is not required to see that they are maintained in a safe and suitable condition, nor does the statute mean to impose that particular duty upon the mine foreman, as supervision of the tracks is not a duty to be implied and it is not essential to the complete performance of any of the acts expressly required of the mine foreman; and under the statute thus considered it therefore remains the imperative duty of the mine operator to see that the tracks, trolley wire, and motors are maintained in a reasonably safe condition.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 942, April, 1915.
Crockett *v.* Keystone Coal & Coke Co. (West Virginia), 84 Southeastern, 948, April, 1915.
See Crockett *v.* Black Wolf Coal & Coke Co. (Alabama), 83 Southeastern, 987.

DANGEROUS TRACK FOR MOTOR.

A mine operator may be liable on the ground of negligence for an injury to a motorman caused by a plank from an old and decayed floor over which the car was required to be run flying up, protruding into the cab and striking the motorman, where the operator maintained its track on an old shop floor, much decayed and of doubtful strength and consistency to operate heavy motors upon even as a temporary makeshift, with the knowledge that mine motors are necessarily built low, are operated close to the track, and loose boards on the track would be liable to fly up and come in on the floor of the cab where the motorman is required to sit and produce such an injury as that complained of.

Yates *v.* Crozer Coal & Coke Co. (West Virginia), 84 Southeastern, 626, p. 629, March, 1915.

DEFECTIVE TRACK—DUTY OF OPERATOR TO REPAIR.

In an action by a miner for injuries caused by his head being caught between a car he was pushing and the wall or rib of the entry, due to an alleged defect in the track, where the question of liability depends on whether the mine operator was to lay the track or whether it was the miner's duty to lay the track, the question of negligence or contributory negligence becomes a question of fact for the jury to determine, and in determining the question the jury has the right to consider the fact that the miner was furnished light rails for the track in his room, while the mine operator laid its part of the track with a heavier rail, and to consider the further fact whether the accident happened while the car was running on the lighter rails or on the heavier rails; and the jury has a right to rely upon the further fact as tending to make the mine owner liable, that the pit boss had examined the defect in the track and promised to have it repaired, as this was a recognition by the operator of the duty upon his part to repair the defect.

El Paso County Land & Fuel Co. *v.* Perdaris (Colorado), 147 Pacific, 675, April, 1915.

PROMISE TO REPAIR.

Where an employee or miner makes a complaint to his employer of a dangerous defect in his place of work, or in the appliances furnished him with which to work, and the employer makes an unconditional promise to repair the defect, the risk of the defect is cast upon the employer until such time as would preclude all reasonable expectation that the promise might be kept, unless the danger from the defect is so imminent that a person of ordinary prudence would not risk injury therefrom; and there can be no distinction in principle, in so far as the liability of an employer is concerned, between an unconditional

promise to repair and a promise to repair on a certain date, or after the happening of a particular event.

El Paso County Land & Fuel Co. *v.* Perdaris (Colorado), 147 Pacific, 675, p. 677, April, 1915.

ESCAPEMENT FROM SHAFT—SIGNALING SYSTEM.

While the doctrine of safe place does not apply in the construction of a mine shaft, whether in excavating or timbering, yet an operator may be liable for a person injured in the sinking of a shaft if he failed to furnish a suitable ladder by means of which the injured employee could get out of the shaft promptly in case of an emergency, and in the failure to furnish a proper and suitable signaling system from the bottom of the shaft or from points in the shaft, so that the bucket or cage could be promptly lowered or hoisted in case of an emergency and where workmen would be required to get out of the shaft quickly in order to be reasonably safe.

Benson *v.* Jones & Laughlin Ore Co. (Michigan), 151 Northwestern, 707, p. 708, March, 1915.

LIABILITY TO INFANT EMPLOYEE.

The liability of an employer for injuries sustained by an infant employee in the course of his employment depends upon the infant's capacity to comprehend and avoid the incidental dangers and whether he is fully advised concerning them; but in the absence of such capacity and intelligence the employer can not escape liability on the ground that the injury to the infant or minor was due to accident or negligence of a fellow servant; and the duty imposed as to dangers incident to the work and not patent to the infant, or the existence of which he is not advised, or by reason of lack of incapacity and experience he can not appreciate or avoid, requires the employer to respond in damages for injuries resulting from such dangers.

Dillon *v.* United States Coal & Coke Co. (West Virginia), 84 Southeastern, 956, p. 959, April, 1915.

DUTY TO PROMULGATE RULES.

Section 493, Code 1913 of Alabama, requires every mine operator to adopt special rules for the government and operation of his mine, covering all the work pertaining thereto in and outside of the same and contemplates the making of a rule regulating the operation of motors and limiting the maximum number of loaded cars that may be hauled on any single trip, as this pertains to the operation of the mine and for the reason that there is more danger in operating a long train than a short one, and a rule on the subject would be a reasonable requirement.

Jaggie *v.* Davis Colliery Co. (West Virginia), 84 Southeastern, 941, p. 943, April, 1915.

DUTY TO PROVIDE SAFE APPLIANCES.

USING DEFECTIVE APPLIANCES—MINER'S CARE.

The rule is that a miner is under no obligation to inspect the instrumentalities provided by the mine operator in order to discover concealed defects or dangers not disclosed by his superficial observation, but the miner may assume that the instrumentalities furnished him by the mine operator are not defective, but suitable for the purposes for which they are furnished, and he is not called upon to seek for and discover hidden defects; but if the defect is obvious and apparent, then he must be held to have assumed the risk of using the defective tool or implement.

Alamo Oil & Refining Co. *v.* Richards (Texas Civil Appeals), 172 Southwestern, 159, p. 160, January, 1915.

DEFECTIVELY EQUIPPED CAGE.

A coal-mining company is liable on the ground of negligence for the death of a miner working at the bottom of a shaft and killed while so working by being struck on the head by a lump of coal that fell down the shaft after being knocked from the bunting of the tipple by an ascending cage in the compartment of the shaft, where the evidence tended to show that the car leaving the lump of coal on the bunting was overloaded and where the evidence tended to prove that the operator had failed to furnish a properly constructed bonnet or other appliance for the protection of the miner from falling coal, and had in that respect failed to exercise reasonable care in furnishing the miner a reasonably safe working place.

Osage Coal & Mining Co. *v.* Miozrany (Oklahoma), 143 Pacific, 185.

DEFECTIVE KICK SWITCH.

A coal-mine operator in Pennsylvania is liable on the ground of negligence for an injury to a driver caused by a collision between two trips of cars, where the collision was the result of a defective appliance known as a kick switch, and the tracks at the location of the switch were in such bad repair that at times the rails broke away from the ties and where there was not sufficient space between the cars and the ribs or walls of coal for the driver to stand in safety, and where the mine foreman and the operator's superintendent both had knowledge of all conditions for at least two weeks before the accident, although the mine was in charge of a certified foreman.

Watson *v.* Monongahela River Consolidated Coal & Coke Co. (Pennsylvania), 93 Atlantic, 625, p. 627, January, 1915.

DEFECTIVE CABLE.

An oil-drilling company is liable on the ground of negligence for using an old, worn, and unsafe "bull rope," and by reason of the defective condition of such rope an employee was injured, where it appears that he had no knowledge of the defective condition of the rope.

Producers' Oil Co. *v.* Eaton (Oklahoma), 143 Pacific, 9, p. 10.

SIMPLE APPLIANCES—DEFECTS IN CONSTRUCTION.

The rule that where a tool or implement is so simple that inspection is not necessary and a man of ordinary intelligence can see any existing defects, then a servant or miner will be charged with an assumption of the risk arising from the use of such a defective implement; but the rule does not apply where the defect is not obvious, but is a latent or concealed defect; but the exception also applies to a ladder furnished by an oil operator, where the injury complained of was occasioned by the giving away of the step or round of a ladder because the nails used by the oil company in constructing the ladder were not large enough for the purpose for which they were used, and where the oil company knew the kind of nails used in fastening the steps on the side of the ladder, and was accordingly charged with the knowledge that the nails were too small, such defective construction being concealed from the employee; and in the absence of observable defects he had the right to assume that his employer had used nails of a size requisite and sufficient to securely fasten the steps of the ladder.

Alamo Oil & Refining Co. *v.* Richards (Texas Civil Appeals), 172 Southwestern, 159, p. 160, January, 1915.

SIMPLE APPLIANCES—REMEDY BY MINERS.

Generally as a means of protection to his miners a coal-mine operator must exercise reasonable care and diligence to keep in repair and good order the machines and appliances used by his miners, and keep himself informed as to their condition by inspection, but a mine operator is not liable for an injury occasioned by a defect arising out of the use of an appliance and susceptible of simple, easy, adequate, and timely remedy by the miners themselves in the absence of proof of his assumption of duty to inspect for such defects and correct them.

Martin *v.* Carter Coal Co. (West Virginia), 84 Southeastern, 574, p. 575, February, 1915.

INSPECTION OF SIMPLE APPLIANCES.

The fact that a coal miner charged with the duty of coupling or loading coal cars and seeing an iron bar or pin, an inch in diameter and approximately 7 inches long, bent slightly out of position, can bend it back to its place by a blow or two from a hammer, pick, or other ordinary implement used about his place of work is a matter of common knowledge of which courts may take judicial notice. The remedy in such case was so simple and easy that presumptively and naturally it would fall within the province and duty of the car-coupler, and provision of a system of inspection for the discovery of such defects would be practically useless, and the law does not impose it in the absence of assumption of such duty by the mine operator.

Martin *v.* Carter Coal Co. (West Virginia), 84 Southeastern, 574, p. 576, February, 1915.

INSPECTION OF MOTOR BY MOTORMAN—LIABILITY FOR INJURY.

A mining corporation owes to its employees the duty to inspect and examine appliances used by them, and this duty is nondelegable; but as corporations can act only through agents they may by general rules impose the duty of inspection upon the employees using the machine or other appliance to the extent that he is competent to make such inspection, and the circumstances afforded him an opportunity of doing so; and the mine operator is not liable for injuries to a motorman caused by defects in the motor where a rule of the mine required the motorman to give his motor daily inspection, and not to run or operate the same unless he believes it to be in an entirely safe condition for service.

Pocahontas Consolidated Collieries Co. *v.* Hairston, 83 Southeastern, 1041, p. 1043 (Virginia), January, 1915.

DUTY TO WARN OR INSTRUCT.

INJURY TO UNINSTRUCTED MINOR EMPLOYEE.

A mine operator is liable in damages for injuries to a minor employee, where the employee was engaged to perform some particular but not hazardous services about a coal tipple, and to carry samples of coal mined from the tipple to the office of the operator for inspection and tests, and where, without instructing him as to any secret or hidden dangers or as to the condition of the track, the mine superintendent directed such minor employee to go in between loaded cars of coal to assist others in pushing them down to the tipple, and where, in doing so, and without any knowledge of the condition of the track and being unable to see because of the presence of the car in front of

him, he caught his foot in a switch and the foot was run over and crushed by the car immediately following the one being pushed.

Dillon *v.* United States Coal & Coke Co. (West Virginia), 84 Southeastern, 956, p. 957, April, 1915.

DUTY TO INSTRUCT INFANT EMPLOYEE.

The law imposes upon a mine operator the duty to warn and instruct an infant employee of latent dangers; and if such infant employee, in obeying the orders of a general superintendent, engages in services other than those regularly assigned to and performed by him, dangers from which such employee does not comprehend or appreciate because unknown to him, but of which the employer knows or by the exercise of reasonable diligence ought to know, and fails so to warn and instruct the youthful employee, the mine operator is liable for resulting injuries to such infant employee, where he is free from negligence directly contributing thereto.

Dillon *v.* United States Coal & Coke Co. (West Virginia), 84 Southeastern, 956, p. 958, April, 1915.

LIABILITY FOR NEGLIGENCE OF FELLOW SERVANT.

WHO ARE FELLOW SERVANTS.

A master or mine operator is not responsible to his servant or miner for the negligence of a fellow workman engaged in the same common employment; and in determining whether an employee through whose negligence defects in the machinery have failed of discovery or repair is a representative of the master or operator in the discharge of his duty to the servant or miner, or is a fellow servant of the latter engaged in a common employment, it is only necessary to determine whether the employee's duty to inspect or repair the apparatus is incidental to his duty to use the apparatus in the common employment; and if so, then he is not intrusted with the employer's duty to his fellow servant, and the employer is not responsible to such fellow servant for his fault; but if the employer has cast a duty of inspection or repair upon the employee who is engaged in using the apparatus in a common employment with his fellow employees, then such employee in the particular duty represents the employer, and the latter is chargeable with his default.

Martin *v.* Carter Coal Co. (West Virginia), 84 Southeastern, 574, p. 576, February, 1915.

TEMPORARY DANGER—VIOLATION OF RULES.

A trammer working in a mine can not recover from the mine operator for an injury caused by the negligence of a blockholer in rolling and throwing stones and materials down an incline in a stope, as

such injury was the result of the negligence of a fellow servant in the same general employment and performing a duty that the operator had a right to delegate to another employee, as ordinarily a mine operator who has provided a safe place, proper appliances, and competent employees fully instructed as to their duties and as to the proper rules and regulations for the safe conduct of the business which are made known to them, is not liable for the negligence of an employee failing to notify a fellow servant of a transitory or temporary danger caused by his acts which will for the moment render the environment unsafe, but which can easily be avoided by due warning, it is the duty to give under rules of which all interested employees have knowledge.

Juntunen *v.* Quincy Mining Co. (Michigan), 151 Northwestern, 571, March, 1915.

OBEYING DIRECTIONS OF FOREMAN.

A foreman having authority to direct miners and the work in a mine is a vice principal, and a miner who follows the judgment of such vice principal is not, as a matter of law, guilty of contributory negligence, unless the situation was so manifestly dangerous that a man of ordinary prudence, in the exercise of due caution, would refuse to obey.

Lindquist *v.* Pacific Coast Coal Co. (Washington), 142 Pacific, 445, p. 447.

PROXIMATE CAUSE.

A mine operator is not liable for an injury to a trammer caused by the negligence of a fellow trammer while attempting to replace a loaded car or buggy upon the track, though the car had been derailed because of the defective condition of the track, as such defective condition of the track was not the proximate cause of the injury, as the jumping of the car from the track in itself did not cause the injury, and the operation of putting the car back on the track was not dangerous if properly conducted, and was an operation distinct in itself.

Sabela *v.* Newport Mining Co. (Michigan), 598, p. 600, March, 1915.

MINE OPERATOR NOT LIABLE.

A mine operator is not liable for an injury caused by the negligence of a fellow miner under the same general employment where the operator has provided a safe place, proper appliances, and competent employees.

Zap *v.* Newport Mining Co. (Michigan), 151 Northwestern, 554, March, 1915.
Mesich *v.* Tamarack Mining Co. (Michigan), 151 Northwestern, 564, p. 565, March, 1915.
Juntunen *v.* Quincy Mining Co. (Michigan), 151 Northwestern, 571, p. 572, March, 1915.
Sabela *v.* Newport Mining Co. (Michigan), 151 Northwestern, 598, March, 1915.

NEGLIGENCE OF SHIFT BOSS.

A trammer in the employ of a mining company can not recover for injuries where he was engaged in taking out ore just blasted for the purpose of putting up timbers, and where by reason of the place looking dangerous he requested the shift boss to make an inspection and where the shift boss did inspect the place and instruct the miner that it was ready and to resume his work, and where the miner on resuming work was shortly afterward injured by the falling of a quantity of roof and ore, on the ground that the negligence of the shift boss was the negligence of a fellow servant.

Zap *v.* Newport Mining Co. (Michigan), 151 Northwestern, 554, March, 1915.

MINER'S WORKING PLACE—SAFE PLACE.

SAFE-PLACE DOCTRINE—APPLICATION.

The "safe-place" doctrine does not apply as a general rule, where the perils to which a miner is subjected are all his own creation, and where, as a result of the miner's work the character of the place is constantly changing; and this rule applies to extracting coal by a miner; and the duty of inspection may, by the contract of hiring or by mutual understanding or custom, in the absence of a statute, be imposed upon the miner; and under such circumstances it is error for a court to instruct a jury, as a matter of law, that the duty of inspection devolved upon the operator.

Eagle Coal Co. *v.* Patrick (Kentucky), 170 Southwestern, 960, p. 962, December, 1914.

Music *v.* Northeast Coal Co. (Kentucky), 170 Southwestern, 971, p. 972, December, 1914.

INSPECTION—PROOF OF CUSTOM.

In an action for damages by a miner injured while working in a mine at a place known as a "glory hole," and where it was the duty of the injured miner, together with others, to remove the broken rock and earth after the drillers drilled the hole and fired the shot, it is proper to prove a rule or custom to the effect that after each shot fired inspection is made of the walls to see that no loose rock remains.

Beatson Copper Co. *v.* Pedrin, 217 Fed., 43.

MINER'S DUTY TO MAKE SAFE.

The making safe of a miner's working place, the place where he is extracting coal, involves three distinct duties: (1) The duty of examining the roof and of determining when protection must be provided against possible falling of the roof and whether ordinary props will be sufficient protection; (2) the duty of placing the props

in position when ordinary props are deemed sufficient protection; (3) the duty of timbering the place when ordinary props have been deemed insufficient protection.

Eagle Coal Co. *v.* Patrick (Kentucky), 170 Southwestern, 960, p. 961, December, 1914.

DUTY TO PROP ROOF—CUSTOM OF MINE.

Ordinarily it is the duty of the mine operator to furnish the necessary props and the duty of the miner to prop his own room; but where a room has been practically worked out and is used as a roadway for loaded cars to reach the entry, the question whether it is the operator's or the miner's duty to prop the roof over the roadway will depend on the custom of the mine.

Stringer *v.* York (Kentucky), 170 Southwestern, 527, November, 1914.

MINER'S FAILURE TO PROP—PROXIMATE CAUSE.

In an action by an injured miner where an injury complained of was not the proximate result of an original defective condition of the roof of an entry, or if the miner failed to do additional propping as the work progressed, then, under the statute of Alabama, he can not recover, though the operator was negligent in failing to remedy the original defect, if the miner's failure to do additional propping as the work progressed contributed with the original defect in the roof of the entry in producing the injury complained of.

Lookout Fuel Co. *v.* Phillips (Alabama), 66 Southern, 946, p. 949, November, 1914.

MINER MAY PRESUME PLACE IS SAFE.

When a mine operator directs a miner to work in a particular place or room, the miner may assume that the operator has performed his duty in furnishing him a reasonably safe place in which to work and he can proceed with his work relying upon this presumption; and it is not the duty of the servant under such circumstances to discover defects, and unless he knows of their existence, or they are patent and obvious to a person of his experience and understanding, he is not precluded from recovering for injuries sustained while working at the direction of the opertaor.

Interstate Coal Co. *v.* Garrard (Kentucky), 173 Southwestern, 767, p. 769, March, 1915.

DEGREE OF CARE REQUIRED OF MINERS.

Where miners are employed in drifting and working into new formations, the degree of care which they will be called upon to exercise for their own protection to avoid injury from a cave or from falling rock will be very much greater than the degree of care imposed

upon them in using the older workings which are or should be in a thoroughly protected condition, but which they have the right to assume are in such condition.

Crabbe *v.* Mammoth Channel Gold Mining Co. (California), 143 Pacific, 714, p. 715.

MINER NOT REQUIRED TO INSPECT.

A miner working in a chamber of a mine 10 feet high, lighted only by the light used by him in mining is not required to inspect the roof of his chamber or station, but under such circumstances he is charged with the duty of exercising only such an amount of care as the law presumes that every person will employ for his own interest and preservation, and the quantity of that care will depend upon and vary with the degree of hazard of the particular occupation.

Crabbe *v.* Mammoth Channel Gold Mining Co. (California), 143 Pacific, 714, p. 715.

MINER OBEYING INSTRUCTIONS.

MINER ACTING UNDER ORDERS OF OPERATOR—DANGEROUS PLACE.

A servant or miner is not called upon to set up his own unaided judgment against that of his superiors, and he may rely upon their advice, and still more upon their orders, and if a master or mine operator orders a servant or miner to do some act which is dangerous, but which could be made less dangerous by the use of special care on the part of the master or operator, the servant or miner has a right to assume that such special care would be taken, and does not take the greater risk upon himself, as in such case the servant's dependent and inferior position is to be considered, and if the master or operator gives him positive orders to go on with the work, under perilous circumstances, the servant or miner may recover for the injury thus incurred, if the work was not obviously so dangerous that any man of ordinary prudence would have obeyed.

Borderland Coal Co. *v.* Small (Kentucky), 170 Southwestern, 8, p. 10, November, 1914.

ASSURANCE OF SAFETY—EFFECT OF CONTRIBUTORY NEGLIGENCE AND ASSUMPTION OF RISK.

Where a miner's working place in a mine is being constantly changed as the work progresses a miner will be held to have assumed the risks incident to his labors, but an exception exists to this rule in that it may happen in the prosecution of the work of mining that the safety of the place will become an object of inquiry calling for an expression of judgment, and under such circumstances a miner who follows the assurance and judgment of the mine operator or his fore-

man will not be charged with an assumption of risk or held to be guilty of contributory negligence as a matter of law, as the assurance is that it is safe to remain in the questioned situation long enough to accomplish the thing to be done, such as removing coal from the particular working place.

Lindquist *v.* Pacific Coast Coal Co. (Washington), 142 Pacific, 445, p. 447.

CONTRIBUTORY NEGLIGENCE—QUESTION OF FACT.

An action by an injured miner against an operator for damages for injuries caused by a fall of rock from the roof is not to be defeated on the ground of contributory negligence as a matter of law, where the miner, after discovering the dangerous condition of the roof, sent for the superintendent of the mine and called attention to the overhanging rock and suggested to the superintendent that it would be safer to drive the entry by pick mining and not by machinery, with its resulting noise and vibration, and where the superintendent sounded the roof with a pick for the purpose of ascertaining whether the rock, a part of which was in plain view, was likely to fall, and who, after thus testing it for the purpose of satisfying himself of its safety, instructed the miner to use the machine and proceed with his work, and where after proceeding to cut the coal by the machine process and had only driven the entry a short distance when the rock fell and injured him, as the question of contributory negligence under such circumstances is one of fact to be determined by the jury.

Moleskey *v.* South Fork Coal Mining Co. (Pennsylvania), 93 Atlantic, 485, p. 486, January, 1915.

RISKS NOT ASSUMED—MINER WORKING UNDER DIRECTION OF OPERATOR.

The rule is that when a place in which a miner is engaged in working is not such as imposes upon the mine operator the full duty of providing a safe place, but is somewhat hazardous or dangerous, although not obviously so, or the danger of continuing is not so apparent that a person of ordinary intelligence would not undertake it, and the miner is assured, in substance or in effect, by the operator or his foreman who is present that it is reasonably safe or that there is no danger, and he is directed by the operator or foreman to continue the work, the miner may recover for injuries received while proceeding with the work, although the risk or hazard in prosecuting the work is as well known to the servant as to the operator or foreman. The reason of this rule is that when the mine operator is present the doctrine of equal knowledge and assumed risk, invoked to relieve the mine operator, should be sparingly applied, and in such case the miner's dependent and inferior position is to be taken into

consideration, as it is not his duty to sit in judgment upon the propriety of the direction or to enter into a discussion with the operator as to the facts upon which his judgment is based, but he may presume that improper orders would not be given and assume that the operator or foreman would not direct him to take risks that were improper. The limitation on this rule is to the effect that the miner can not recover where the risk was such that a person of ordinary prudence, situated as the miner was, would not have undertaken it.

Northeast Coal Co. *v.* White Oak Coal Co. (Kentucky), 174 Southwestern, 732, p. 733, March, 1915.

OBEYING INSTRUCTIONS OF SUPERIOR.

A miner is not called upon to set up his own unaided judgment against that of the superintendent of a mine, and he may rely upon the advice and skill and follow the orders of the superintendent, notwithstanding any misgivings of his own, as under such circumstances the miner's dependent and inferior position is to be taken into consideration; and in such case, if the superintendent gives him positive orders to go on with the work under perilous conditions and circumstances, the miner may recover for an injury thus incurred if the work was not inevitably and imminently dangerous; and the question as to whether the work was inevitably and imminently dangerous may be a question of fact.

Moleskey *v.* South Fork Coal Mining Co. (Pennsylvania), 93 Atlantic, 485, p. 486, January, 1915.

DIRECTING WORK IN DANGEROUS PLACE.

The rule that a mine operator is not required to furnish a safe place where the miner himself is engaged in making the place does not apply where the operator by its foreman directed a young and inexperienced miner to go into a room where he had not been working to cut down the coal with a mining machine run by electricity, and where the miner was injured by a fall of slate while operating the machine cutting the coal, and where the operator knew that the place was not safe.

Interstate Coal Co. *v.* Garrard (Kentucky), 173 Southwestern, 767, p. 768, March, 1915.

MINER MAY RELY ON JUDGMENT OF SUPERIOR.

The rule that a mine operator is required to furnish a miner a safe working place does not apply where the miner himself is engaged in making the place; and the rule may apply to the work of mining out or exhausting pillars in a mine which is from its very nature more or less hazardous; but the rule does not apply where miners, having removed the greater part of the coal from a pillar and having

left only one or two small pillars as natural supports, which the miners intended to leave as supports, and on reporting such fact to the foreman, were assured that it was safe to remove the pillars and stumps and were thereupon directed by him to do so, as the miners had the right to rely upon the assurance of the foreman if the danger was not so obvious and imminent that no reasonably prudent man would undertake it even under the positive orders of the foreman or operator.

Northeast Coal Co. *v.* White Oak Coal Co. (Kentucky), 174 Southwestern, 732, March, 1915.

ASSURANCES OF SAFETY.

In directing an inexperienced miner to operate a machine for cutting coal in a room or entry where he had not been working, it is not necessary that the operator should use words which in terms expressly or directly assured the miner that the place where he was directed to work is a reasonably safe place; but it is sufficient if the acts of the operator under all the circumstances amount to an assurance that the place where he directs the miner to work is a reasonably safe place.

Interstate Coal Co. *v.* Garrard (Kentucky), 173 Southwestern, 767, p. 768, March, 1915.

CONTRIBUTORY NEGLIGENCE OF MINER.

CONTRIBUTORY NEGLIGENCE—ASSUMPTION OF RISK.

Neither the defense of contributory negligence nor the defense of assumption of risk can arise unless the defendant in the action—a mine operator—has been guilty of negligence which, but for want of both of these defenses, would render the operator liable for damages to an injured miner, as in the absence of such negligence there is nothing against which to make such a defense; but if there is evidence from which a jury may find the operator guilty of such negligence, then either of these defenses, if it exists in fact, is available to the defendant operator in order to defeat a recovery.

Osage Coal & Mining Co. *v.* Sperra (Oklahoma), 142 Pacific, 1040, p. 1043.

CLIMBING ON MOVING CAR.

A mine foreman or tipple boss who had been in charge of the tipple and the work connected therewith continuously for a year and a half, and whose duty it was to inspect its conditions and report defects, and who was also intrusted with the duty of supervising and directing the moving and loading of coal cars at the tipple, and of directing the other employees of the mine operator working at the tipple, is guilty of such contributory negligence as will prevent a

recovery for his death caused while attempting to climb on a moving car and in doing so placed one foot in the stirrup at the front and corner of the car and the other on the car beam, and while in this position the car in passing under the tipple caused his body to come in contact with a tipple post or brace and be caught between it and the moving car, which crushed and killed him.

Louisa Coal Co. *v.* Hammond (Kentucky), 169 Southwestern, 709, p. 711, October, 1914.

MINER REMOVING PROPS.

Where the roof over a roadway was securely propped by the mine operator and the miner in removing the tracks knocked the props down, he can not recover for a subsequent injury caused by the falling of the roof of the roadway on the ground that it was not securely propped.

Stringer *v.* York (Kentucky), 170 Southwestern, 527, p. 528, November, 1914.

TRACK MAN OPERATING MOTOR.

There can be no recovery for the death of a track man employed in a mine where he voluntarily took charge of the motor hauling loaded cars out of the mine and where he operated the motor with the trolley in front, and while being thus operated the trolley jumped from the wire and struck against the roof of the mine and the driver was either hit by the pole or in getting out of the way of the pole was caught between the motor and the timbers, and where it appeared that his use of the motor was in violation of a rule of the mine.

North Jellico Coal Co. *v.* Disney (Kentucky), 171 Southwestern, 192, December, 1914.

STOPPING IN FRONT OF MOVING CAR.

A miner working in a mine, who without looking stopped in front of a moving car and was struck and injured thereby, is guilty of such contributory negligence as will prevent a recovery for such injury.

Ranzier *v.*. Monongahela River Consolidated Coal & Coke Co. (Pennsylvania), 93 Atlantic, 501, p. 502, January, 1915.

RIDING ON CAR IN VIOLATION OF RULE.

In an action by a licensee for an injury while riding on a tramcar in a mine, where the complaint averred that the plaintiff was, when injured, in a place where he had a lawful right to be, imposes upon the plaintiff the unnecessary burden of vindicating his right to be on the tramcar at the time he was injured, and the defendant, the mine operator, may, under the general issue prove the existence of a rule prohibiting employees riding on the cars.

Woodward Iron Co. *v.* Finley (Alabama), 66 Southern, 587, p. 588, November, 1914.

VOLUNTARY EXPOSURE TO DANGER—PROXIMATE CAUSE OF INJURY.

A miner working in a coal mine is bound to take notice that it is a dangerous occupation and it is his duty to use ordinary care and prudence to observe the rules and customs of the mining company employing him, not in violation of law, to prevent an accident or injury; and where a miner is informed or has notice of danger which threatens injury to himself, or where he voluntarily puts himself in a dangerous place, or where, by ordinary observation and care, he could avoid such danger by reasonable exertion, and fails to do so, he is guilty of contributory negligence. The term "contributory negligence," as here used, must be such negligence on the part of the miner as helped to produce the injury complained of, and in order for such contributory negligence to be sufficient to prevent the miner from recovering it must have been the proximate cause of his injury.

Rock Island Coal Mining Co. *v.* Davis (Oklahoma), 144 Pacific, 600, p. 606, December, 1914.

MINER PLACING HIMSELF IN DANGEROUS POSITION.

A mine operator is not liable for injuries to an engineer employed to operate the engine used for hoisting ores from its mine, where, while he was cleaning the engine, he was injured by steam escaping from the boilers into the cylinders and starting the engine suddenly, where it appears that the engineer was thoroughly experienced in handling the engine, knew its condition and that its throttle valve had leaked steam into its cylinders for years, and that the defect was getting worse, and where he knew from constant observation and practical experience that the engine was liable to start any moment even with the throttle valve closed and the cylinder cocks open, and where in discharging his duties in respect to cleaning the engine between car trips it was wholly unnecessary for him to be in the position in which he was and where he placed himself and the parts of his body in such dangerous position thoughtlessly and in utter disregard of the obvious and imminent danger.

Sloss-Sheffield Steel & Iron Co. *v.* Reid (Alabama), 68 Southern, 136, p. 137, February, 1915.

SETTING CAR IN DANGEROUS PLACE.

A miner creates his own dangerous working place and is guilty of such contributory negligence as will prevent a recovery where while laying a track in a room neck he pushed a car back a short distance in the neck, and where the injury complained of was caused by the car being struck and moved down upon him by a trip of cars over the main track, and where it also appeared that the injured miner knew that the driver would be along soon, and where the driver had no notice that such car in the neck was not in the clear.

Jellico Coal Mining Co. *v.* Gothard (Kentucky), 170 Southwestern, 649, November, 1914.

VIOLATION OF RULES.

A coal operator may prescribe and promulgate reasonable rules for the performance of the respective duties of the miners, and where such rules have been properly established the operator is entitled to a compliance therewith by the miners; but if the miner elects to perform his duty by a method known to him to be dangerous, and in violation of the established rules, the operator will not be liable for an injury to the miner, whether the danger is obvious or not, and the contributory negligence of the miner in the violation of the rule may preclude his recovery, but to do so the violation of the rule must contribute to and cause the injury complained of.

San Bois Coal Co. *v.* Resetz (Oklahoma), 143 Pacific, 46, p. 50.

VIOLATION OF RULE—PROXIMATE CAUSE.

The violation of a rule by a miner working in a mine will not defeat a recovery in an action for his death on the ground of the alleged negligence of the coal mine operator, unless it is made to appear that such violation of the rule was the proximate cause of the death of the miner.

San Bois Coal Co. *v.* Resetz (Oklahoma), 143 Pacific, 46, p. 50.

QUESTION OF FACT AS TO EXISTENCE OF RULE.

It is a question of fact for a jury as to whether or not a miner was guilty of contributory negligence in going down into a mine in a car, upon the occasion when he lost his life, in violation of a rule of the mine operator prohibiting miners from doing so, where the evidence is conflicting as to whether or not such a rule existed and whether or not it applied to persons or miners going down in cars with timbers to scotch when the cable was being changed.

Southern Iron & Steel Co. *v.* Boston (Alabama), 66 Southern, 684, p. 686, November, 1914.

CONTRIBUTORY NEGLIGENCE A QUESTION OF FACT.

The statute of Michigan provides that no male under the age of 18 years shall be employed in any employment which may be considered dangerous to life and limb; and in an action for the death of a drill boy under 16 years of age employed in a mine, the court can not say as a matter of law that the deceased was guilty of contributory negligence, or that his parents, the beneficiaries, were guilty of contributory negligence, where they had no knowledge of the nature or the dangerous character of the work, but the question of contributory negligence under such circumstances is a question of fact to be determined by the jury.

Paskvan *v.* Allouez Mining Co. (Michigan), 152 Northwestern, 82, p. 84, April, 1915.

Where a trapper in charge of a trapdoor in a haulageway of a mine, whose duty it was to open the trapdoor for the motor and cars, was run over and killed by the motor, there is no presumption of negligence on the part of the decedent, nor is there any presumption of negligence on the part of the motorman; but the negligence of either, or both, must be established by the evidence or by circumstances from which such negligence could properly be inferred and such questions are always properly left to the jury to determine.

Linard *v.* Interstate Coal Co. (Kentucky), 169 Southwestern, 1006, p. 1008, October, 1914.

ADMISSIONS OF NEGLIGENCE—EFFECT.

A miner is not necessarily and conclusively bound by statements made by him immediately after receiving an injury in a coal mine to the effect that the injury was his own fault and it was a foolish thing for him to go to cutting through the vein with the coal in the situation it was, and that he had not struck but two or three blows until it all fell and broke his leg, as such admissions are not admissions of a fact but of a conclusion of law; but the question as to whether he was entitled to recover is not whether there was some risk or danger attending the transaction of his work, but whether it was so apparent and imminent that no reasonable man would have undertaken it, and his statements made do not go to that extent, and especially where the fall of the coal was due to the fact that the mine operator had not furnished the miner props as required by the statute; and the statements made did not support a conclusive inference that the danger was so great that no reasonable man would have encountered it.

Runyan *v.* Marceline Coal & Mining Co. (Missouri Appeals), 172 Southwestern, 1165, p. 1167.

PLEADING CONTRIBUTORY NEGLIGENCE—SUFFICIENCY OF ANSWER.

In an action by an injured miner for damages on the ground of the alleged negligence of the mine operator in that the operator failed to furnish the miner a safe place in which to work, an answer to the complaint is sufficient where it avers that the plaintiff himself was guilty of negligence which proximately contributed to the injuries complained of, in that the plaintiff while knowing that a rock in the room in which he was working in the mine was loose and liable to fall, and that if it should fall while he was under or near the same he was likely to be injured, and he negligently worked under or dangerously near such rock and as a proximate consequence thereof the rock fell upon him causing the injuries for which he sues.

Alabama Fuel & Iron Co. *v.* Benenante (Alabama), 66 Southern, 942, p. 943, November, 1914.

PLEADING CONTRIBUTORY NEGLIGENCE—PROXIMATE CAUSE.

In an action by a miner for damages for injuries caused by the alleged negligence of the mine operator in failieg to furnish the miner a safe working place, an answer which avers that the miner was informed of the dangerous condition of the rock and the roof of the room at the place at which the injury occurred and knowing that the same was liable to fall at any moment and likely to injure him, yet he negligently placed himself under the same and was injured, is insufficient in that it fails to aver that the negligence on the part of the miner proximately contributed to his injury.

Alabama Fuel & Iron Co. *v.* Benenante (Alabama), 66 Southern, 942, p. 943, November, 1914.

WANT OF KNOWLEDGE OF DANGER.

Defects in the roof of a mine which might be apparent to the eye of a competent inspector might not have any significance to a laborer or employee who has had no experience in this special employment; and under such circumstances it would be unreasonable to charge a miner with contributory negligence merely because he sees defects, unless a reasonably prudent man would under like circumstances, have known or comprehended the risk which these defects indicate. The rule is that the dangers and not the defects alone must be so obvious that a reasonably prudent person would have avoided them, in order to charge the miner with contributory negligence.

Interstate Coal Co. *v.* Garrard (Kentucky), 173 Southwestern, 767, p. 769, March, 1915.

FREEDOM FROM CONTRIBUTORY NEGLIGENCE.

CONTRIBUTORY NEGLIGENCE AT OTHER TIMES.

In an action for damages for the death of a miner by the alleged negligence of a mine operator, it is no defense to show that the deceased miner was guilty of negligence and was habitually negligent at times other than that at which he lost his life; and it was no defense where there was an entire absence of proof of contributory negligence at the particular time at which the miner lost his life, as evidence of habitual negligence as to past occurrences is inadmissible to prove contributory negligence on the particular occasion under inquiry.

Great Western Coal & Coke Co. *v.* McMahan (Oklahoma), 143 Pacific, 23, p. 24.
Great Western Coal & Coke Co. *v.* Cunningham (Oklahoma), 143 Pacific, 26.
Great Western Coal & Coke Co. *v.* Coffman (Oklahoma), 143 Pacific, 30.
Great Western Coal & Coke Co. *v.* Boyd (Oklahoma), 143 Pacific, 36.
Great Western Coal & Coke Co. *v.* Belcher (Oklahoma), 143 Pacific, 36.

WANT OF KNOWLEDGE OF DANGER OF DEFECTIVE ROOF.

A father in an action for the death of a son, working with him in a mine, is not to be charged with such contributory negligence as will prevent him from recovering, where it does not appear that either the father or the son, who were working together, knew the actual condition of the roof and of the rock that fell, causing the death of the son.

Rocky Mountain Fuel Co. *v.* Kobaics (Colorado), 144 Pacific, 863, p. 866, December, 1914.

MINER ACTING UNDER SUDDEN FRIGHT.

A miner is not, as a matter of law, to be charged with contributory negligence where it appears that he was suddenly frightened by the presence of a mule drawing a car in a haulage way and in order to escape injury by the mule and the car stepped on the side of the track on which there was the least room for the car to pass and in so doing came in contact with an uninsulated electric wire, as the condition of the wire under the circumstances must be regarded as the proximate cause of the injury.

Hazzard *v.* Consolidated Coal Co. (Michigan), 149 Northwestern, 991, p. 992, December, 1914.

ACTING UNDER SUDDEN IMPULSE—PROXIMATE CAUSE OF INJURY.

An employee assisting in drilling an oil well is not, as a matter of law, to be charged with contributory negligence in suddenly applying the brake on the bull wheel in order to stop the descending sucker rods with such force and suddenness as to break the bull rope, where he was directed by the operator of the machinery to apply the brake and where there was a necessity for quick action, and where the injury was caused by the breaking of the bull rope, due to its old, worn, and defective condition, and of which defective condition the employee had no knowledge, as in such case the defective condition of the rope, and not the application of the brake to the machinery, was the proximate cause of the injury.

Producers' Oil Co. *v.* Eaton (Oklahoma), 143 Pacific, 9, p. 10.

RISKS ASSUMED.

KNOWLEDGE OF DANGERS.

A coal-mine operator is not absolutely and unqualifiedly bound to furnish, and is not absolutely and unqualifiedly liable for failure to furnish, a miner a reasonably safe place in which to work and reasonably safe tools and appliances with which to work; but the operator's duty to do so is subject to the qualification that the miner assumes the usual, ordinary, and known risks arising from the operator's fail-

ure to do so, and the risks thus assumed by the miner may have resulted from what would otherwise have been actionable negligence on the part of the operator.

Osage Coal & Mining Co. *v.* Sperra (Oklahoma), 142 Pacific, 1040, p. 1043.

QUESTION OF FACT.

Under the constitution of Oklahoma the defense of contributory negligence or assumption of risk shall in all cases be a question of fact and shall in all cases be left to the jury for determination.

Osage Coal & Mining Co. *v.* Sperra (Oklahoma), 142 Pacific, 1040, p. 1043.

DANGER FROM FALLING ROCK.

A member of a repair crew making repairs in a shaft for the purpose of making the shaft safe, which had been rendered unsafe by the crushing in of the shaft, can not recover for injuries caused by falling rock in the progress of the work, as the danger from such falling rock and material was assumed, where it was the duty of the injured employee and that of his fellow laborers to bore down and carry away such rock as was loose and dangerous in the vicinity of their labor and where it was well known to him that the place was not safe.

Juntunen *v.* Quincy Mining Co. (Michigan), 151 Northwestern, 571, p. 572, March, 1915.

QUESTION OF FACT.

It is a question of fact for a jury to determine whether or not a miner assumed the risk of going down in a car loaded with timbers instead of walking down, where the evidence is such that the jury may infer that the usual and customary way was to take the timber down and come with it in the car, and though the proof may show that it was safer to walk down, but does not conclusively show that it was dangerous or hazardous to ride down on the car.

Southern Iron & Steel Co. *v.* Boston (Alabama), 66 Southern, 684, p. 686, November, 1914.

KNOWLEDGE OF WANT OF SAFETY DEVICES.

A miner who undertakes to work in a mine with knowledge of the absence of a derailing switch at the mouth of the mine and without protection that such switch might give him, and who relies upon the care of his fellow workmen and the efficacy of a wooden block used to guard the mouth of the mine against runaway cars, can not recover against the mine owner for injuries caused by a runaway car, and especially so where the mine was operated by an independent contractor by whom the injured employee was employed.

Connors Weyman Steel Co. *v.* Kilgore (Alabama), 66 Southern, 609, p. 612, November, 1914.

EMPLOYEE ENGAGED IN SINKING A SHAFT.

An employee entering the employment of a mine operator and engaged in the work of sinking a shaft voluntarily assumes all the obvious risks and dangers ordinarily incident to such mining work; and it is his duty to familiarize himself with the imminence of the danger and he is taken to have assumed all the ordinary risks which are incidental to the work of sinking the shaft of which he actually knows, and also such risks which every man, using ordinary prudence and care in the performance of such work, would be bound to learn and know of such result of doing such work, and for an injury resulting from any such obvious or usual risk or danger, there can be no recovery.

Benson *v.* Jones & Laughlin Ore Co. (Michigan), 151 Northwestern, 707, p. 708, March, 1915.

FAILURE TO PROP ROOF.

Under the custom of a mine in Kentucky where it is the duty of a miner to set such props as are necessary and to examine the roof when he blows down the coal, and where danger is created by the miner in the progress of his work, the miner assumes the risk resulting from the change made in the place of work by him in the ordinary progress of the work.

Music *v.* Northeast Coal Co. (Kentucky), 170 Southwestern, 971, p. 972, December, 1914.

EFFECT OF CONTRACT OF EMPLOYMENT.

Assumption of risk is a term of a contract of employment, either express or implied from the circumstances of the employment, and by which the servant agrees that dangers of injury obviously incident to the discharge of his duties, shall be at his risk.

Osage Coal & Mining Co. *v.* Sperra (Oklahoma), 142 Pacific, 1040, p. 1043.

DEATH FROM ASSUMED RISKS—NO RECOVERY.

A miner employed in a mine does not assume all the risks of his employment, but only such as are ordinarily incidental to the employment, and whether a miner assumes the risk or not is a question for the jury, where men of ordinary judgment on the facts might reasonably differ in opinion. Accordingly, where an employee entered the service of a mine operator as a trapper he assumed all the ordinary risks incident to his employment as trapper, and if his death was caused by the ordinary risks of the employment which he had assumed, and not by reason of the negligence of the operator, then there can be no recovery for the death of such trapper.

Linard *v.* Interstate Coal Co. (Kentucky), 169 Southwestern, 1006, p. 1008, October, 1914.

MINER ACTING AS REPRESENTATIVE OF OPERATOR.

A servant or miner is not ordinarily required to make a minute or detailed examination of the place where the master or mine operator puts him to work, nor to take notice of any defects which would not be apparent to one who usually has neither time nor opportunity for more than a casual, hurried glance at the place of work or the instrumentalities, and under such circumstances he may rely on the master or operator having adequately discharged his primary duty of using ordinary care to make the place of work and instrumentalities of work reasonably safe for his use; but where the servant, or miner, as the representative of the master or operator, is himself in control of the place of work, or instrumentalities for doing it, and the manner of its performance, and if he himself undertakes its performance, he then assumes not only the risks or dangers that are obvious but also such as ordinary care on his part in inspecting the place or instrumentalities of work before beginning it, could have enabled him to discover it.

Louisa Coal Co. *v.* Hammond (Kentucky), 169 Southwestern, 709, p. 711, October, 1914.

VIOLATION OF RULE.

A coal miner injured in attempting to remove a coal-cutting machine by a method positively prohibited by a rule of the operator because of its danger, can not recover damages for the injury, and especially where it appears that the injury would not have occurred if the machine had been moved according to the rule and directions of the mine operator.

Fluhart Collieries Co. *v.* Meets (Kentucky), 169 Southwestern, 686, p. 687, October, 1914.

RISKS NOT ASSUMED.

DEFECTIVE CONSTRUCTION OF SIMPLE APPLIANCE.

An employee of an oil company is not charged with the assumption of risk in the use of a defective ladder, where the defect consisted in the smallness of the nails used to fasten the steps on the sides of the ladder and where the defect was not so apparent or obvious as to charge him with knowledge; and an employee is not charged with the duty of inspection, nor even the exercise of ordinary care, the defect being one of construction, and being the case of a ladder constructed by the employer in a defective manner and which defects were hidden from the employee and which defects in construction caused the injury for which the employee sues.

Alamo Oil & Refining Co. *v.* Richards (Texas Civil Appeals), 172 Southwestern, 159, p. 160, January, 1915.

FAILURE OF OPERATOR TO FURNISH MINER SAFE PLACE.

Where the negligence of a coal-mine operator in failing to provide a miner with a reasonably safe place to work or reasonably safe machinery or tools with which to perform the work, is the proximate cause of an injury sustained by the miner, the miner will not be held to have assumed any risk that would not, by the use of ordinary care, have been obvious to a person of ordinary intelligence in a like situation.

Fluhart Collieries Co. *v.* Meets (Kentucky), 169 Southwestern, 686, p. 689, October, 1914.

ELECTRIC WIRES IN HAULAGEWAY.

A miner working in a coal mine assumes the risk of those dangers which he actually knows or of which he ought to have known, but as a matter of law, he is not charged with the assumption of the risk of danger from electric wires hanging in a passageway through which he was required to pass and where there was a gap left in the insulation of the wire and where he came in contact with the wire and received the injury in an attempt to escape from a mule hauling a coal car that came upon him suddenly and unexpectedly, and especially when he testified that he had no knowledge of the defective insulation or covering of the wire.

Hazzard *v.* Consolidated Coal Co. (Michigan), 149 Northwestern, 991, p. 992, December, 1914.

NEGLIGENCE OF OPERATOR NOT ASSUMED.

While a servant or miner by implication agrees that he will undertake the ordinary risks incident to the services in which he is to be engaged, yet he does not assume any risk which may be obviated by the exercise of reasonable care on the master's part, and any failure on the part of a mine operator to observe, for the protection of his miners, that reasonable degree of care which the circumstances of the particular case justly demand, is actionable negligence and is not within the influence of the doctrine of assumed risk.

Darby Coal Mining Co. *v.* Shoop (Virginia), 83 Southeastern, 412, p. 415, November, 1914.

TWO WAYS OF DISCHARGING A DUTY.

The rule requiring a miner or other employee to select the safe or less dangerous way of performing service, does not apply although one way is safer than the other, if the other way is not obviously or necessarily dangerous, and especially if a custom of a mine permits the performance of the service in the more dangerous way.

Southern Steel & Iron Co. *v.* Boston (Alabama), 66 Southern, 684, p. 686, November, 1914.

EFFECT OF PROMISE TO REPAIR.

Assumption of risk is a matter of implied contract and an employee may be held to have assumed a risk, though his own act in proceeding in the face of danger would not constitute negligence on his part; but the employer's promise to repair a defect operates as a suspension of the employee's implied contract to bear the risk, and puts the obligation on the operator to bear the risk during the period covered by his promise. Accordingly where a pit boss promised a miner on Saturday to have a defective track repaired before Monday, the miner on returning to his work on Monday morning was justified in relying upon the promise of the pit boss and in believing that the defect had been repaired as promised.

El Paso County Land & Fuel Co. *v.* Perdaris (Colorado), 147 Pacific, 675, p. 677, April, 1915.

CONTRACTS RELATING TO OPERATIONS.

RECOVERY FOR BREACH OF CONTRACT—APPEAL AND JURISDICTION.

The Court of Appeals of Kentucky has no jurisdiction on an appeal from a judgment for less than $200, a recovery for coal mined by the plaintiff, where the title to the land is not in controversy.

Burk Hollow Coal Co. *v.* Lawson (Kentucky), 169 Southwestern, 695, October, 1914.

PURCHASE OF MINING PROPERTY—RIGHT TO RESCIND FOR FRAUD.

A contract for the sale of mining property may be rescinded on the ground of fraud and misrepresentation where the fraudulent representation consisted in stating that the mine could be operated on a basis of a 20 per cent royalty, when in fact the seller while operating under a 20 per cent royalty with the landlord, was bound by a contract with other parties by which he was compelled to pay an additional 10 per cent royalty, and which fact was fraudulently concealed from the purchaser; and the right to rescind can not be defeated from the fact that the purchaser made an independent investigation and ascertained from the landlord that the royalty was on the 20 per cent basis, when in fact the purchaser had no knowledge as to the additional 10 per cent royalty.

Glass *v.* Templeton (Missouri Appeals), 170 Southwestern, 665, p. 666, November, 1914.

SUIT FOR ROYALTY—EQUITY JURISDICTION.

An action growing out of a controversy over the right to oil royalties is never more than a mere money demand and shows no ground of equity cognizance, for to justify the interposition of a court of equity there must be something more than a mere claim or

demand, there must appear some equity relief to such claim or demand, something remedial to the plaintiff that the law does not give.

Peterson v. Smith (West Vriginia), 84 Southeastern, 250, January, 1915.

CONSTRUCTION OF CONTRACT TO MINE COAL—RIGHTS AND LIABILITIES.

A contract to mine coal whereby the mine owner might, in his discretion, control the daily tonnage the miner might remove, and thereby indefinitely prolong the process of removing the coal, is sufficient to impel the conclusion that the contractual intent was for an employment merely, and determinable, without breach, at the will of either party, as otherwise the miner would be obligated to the wholly unreasonable extent of indefinitely holding himself in readiness to mine whenever the mine owner so desired, or to regulate the amount of coal he would remove according to the owner's ungoverned desires; but if the owner's only right was to suspend mining by the miner because there were no orders acceptable to and then accepted by the owner, and there were in fact accepted orders on the part of the owner, and the owner then prevented the miner from mining the coal, this would be a breach of the contract.

Pratt Consolidated Coal Co. v. Short (Alabama), 68 Southern, 63, p. 66, February, 1915.

CONTRACTUAL RELATION—VALIDITY OF CONTRACT.

In an action for the breach of a contract to mine coal entered into by a mine owner and a miner, an averment of the contractual right of the miner to mine all the coal, within a reasonable time, in a certain area or territory in the mine of the owner, and the correlative obligation on the owner to accept and pay a stipulated sum per ton for the coal so mined and delivered by the miner at a designated tipple of the owner, is sufficient to show a contractual relation and to relieve the contract of any uncertainty and indefiniteness in respect of right and obligation as would render the contract invalid.

Pratt Consolidated Coal Co. v. Short (Alabama), 68 Southern, 63, p. 65, February, 1915.

CONTRACT TO MINE DETERMINABLE AT WILL OF EITHER PARTY.

A contract between a mine owner and an independent contractor by which the latter is given the right to mine, for a stipulated sum per ton, all the coal in a defined area in one of the owner's mine entries, but which contains no provision for the time or period in or during which the right to mine the coal is assured to such contractor, and where it appears that the actual right of the contractor to mine the coal depended upon the condition whether the mine was operated

by the owner, the contract is determinable at the will of either party, and the mine owner is not liable for a breach of the contract.

Christian *v.* Stith Coal Co. (Alabama), 66 Southern, 641, November, 1914.

UNILATERAL CONTRACT—EFFECT OF OPERATIONS.

A contract to mine coal by which the miner was given the excusive right to mine all the coal in a certain definite territory can not be held void because it is unilateral, and contains no stipulation or promise obligating the miner to mine all the coal in the stated territory, where the miner had proceeded with the work and made an outlay in accordance with the engagement, under the rule that if the party in whose favor a unilateral promise is made accepts it, performs or does any act in recognition of its implied though unexpressed consideration, this supplies the element of mutuality and makes the contract enforceable.

Pratt Consolidated Coal Co. *v.* Short (Alabama), 68 Southern, 63, p. 67, February, 1915.

TIME OF PERFORMANCE—PRESUMPTION.

Where a contract to mine all the coal in a certain area or territory is silent as to the time in which it should be performed by the miner, the law presumes that the parties intended it should be performed by them within a reasonable time.

Pratt Consolidated Coal Co. *v.* Short (Alabama), 68 Southern, 63, p. 67, February, 1915.

CONTRACT WITH BROKERS TO SELL OUTPUT OF MINE.

A contract by which a coal-mining company agreed that certain brokers should have the exclusive sale of its output of coal and coke so long as the services of the brokers were satisfactory and they demonstrated their ability to handle the same and by which the coal company agreed that no one else should quote on its product, and by which the brokers agreed to keep the coal company supplied with orders at good prices, to bill and collect for the coal and coke sold, and to guarantee payment of all accounts, is not lacking in mutuality and the contract is not terminable at the will of either party, and the fact that the coal company made a sale of coal in violation of the terms of the contract was not itself a revocation of the agency and for which the coal company would be liable to the brokers for their commission, and to relieve the coal company of this liability it must give notice of its intention to terminate the contract before it could relieve itself from liability for sales of coal made by it.

Elkhorn Consolidated Coal & Coke Co. *v.* Eaton, Rhodes & Co. (Kentucky), 173 Southwestern, 798, p. 800, March, 1915.

INDEPENDENT CONTRACTOR.

INJURY TO MINER—OPERATOR AN INDEPENDENT CONTRACTOR.

A coal miner who is employed by another miner under contract with the owner and operator of the coal mine must be regarded as a servant of the coal-mine operator to the extent that he is entitled to the protection afforded by statute imposing certain duties upon the coal-mine operator; and the operator can not be permitted to escape liability on the pretense that the alleged employer is an independent contractor.

Lehigh Valley Coal Co. *v.* Yensavage, 218 Fed., 547, p. 552.

LIABILITY OF OPERATOR FOR INJURY TO EMPLOYEE OF THIRD PERSON.

A mine operator owes to a servant or an employee of an independent contractor, mining coal by the ton, the duty of exercising ordinary care to give such employee a reasonably safe place to work, and the mine operator is not relieved from this duty by delegating it to others or to independent contractors; and the negligence of the independent contractor under such circumstances is the negligence of the mine operator and for which it must respond in damages to the injured employee.

Bon Jellico Coal Co. *v.* Murphy (Kentucky), 171 Southwestern, 160, p. 161, December, 1914.

MINE OWNER'S DUTY TO CONTRACTOR'S EMPLOYEES.

A mine owner who employs an independent contractor to operate his mine and who thereby impliedly invites such contractor and his employees to enter and use the mining premises, is liable to them for personal injuries resulting from any condition of the premises which is inherently dangerous, if the contractor and his employees have no knowledge or notice of such condition; but no duty devolves upon the mine owner by reason of this relationship to furnish any part of the equipment necessary or suitable for the operation of the mine, and there is no duty on him to select and install such safety appliances for tracks, trams, or machinery as would render their negligent operation by the contractor or his employees less likely to result in injury to the latter; and in the absence of any agreement to this effect the mine owner owes no duty to an employee of such independent contractor to provide a derailing switch near the mouth of the mine.

Connors Weyman Steel Co. *v.* Kilgore (Alabama), 66 Southern, 609, p. 610, November, 1914.

CONTRACT FOR OPERATING MINE—DUTIES IMPOSED ON OWNER.

Provisions in a contract between a mine owner and an independent contractor for the operation of a mine to the effect that the mining shall be done in accordance with the best methods and with the min-

ing laws of the State, and to the satisfaction of the owner and his engineer, are obviously for the protection of the mine and the mine owner and not for the benefit of employees of such independent contractor, except as they may be operated incidently in that behalf, and does not impose upon the owner the duty of furnishing a particular device to insure the safety of the contractor's employees.

Connors Weyman Steel Co. *v.* Kilgore (Alabama), 66 Southern, 609, p. 611, November, 1914.

MASTER AND SERVANT—EMPLOYEE OF INDEPENDENT CONTRACTOR.

Where a head miner, as is common, employed assistants and helpers, who stand in the same relation to the mine owner and operator and the mine boss as the head miner stands, it is the duty of the mine owner and operator to protect such helpers in every way and to the same extent as he should protect the head miner; but the head miner or workman in such case is not an independent contractor, but a servant whose position depends on the amount of coal he gets out and in such case the amount paid his helpers by him is taken out of his pay, and where both are subject to the control of the mine owner and operator they are equally his servants.

Borderland Coal Co. *v.* Small (Kentucky), 170 Southwestern, 8, p. 9, November, 1914.

PARTNERSHIP AGREEMENTS.

MINING PARTNERSHIPS—WHAT CONSTITUTES.

A mining partnership exists where the several owners of a mine cooperate in the working of a mine; and a mining partnership may exist as well where the parties have a common interest merely in the working of a mine or in carrying on mining operations as where they own the mine itself; and when two or more persons acquire mining properties solely or principally for the purpose of extracting ores, in the absence of an express intention to enter into joint commercial partnership in the operation of a mine, the relation existing between them in the transaction of other common business is a mining partnership.

Lamont *v.* Reynolds (Colorado), 144 Pacific, 1131, December, 1914.

PROMOTERS NOT PARTNERS.

The fact that two persons joined with others as promoters in the organization of a corporation to purchase mines and mining properties and the fact that two of them subsequently executed a declaration of trust by which they joined their holdings of stocks and bonds, is not sufficient to constitute a partnership authorizing one to bind the other by contract or to make one liable to account to the other for profits received.

Ringolsky *v.* Maul L. Mining Co. (Missouri), 171 Southwestern, 56, p. 59, December, 1914.

OPERATING LEASE BY JOINT OWNERS.

Joint owners of an oil and gas lease who worked the lease together though not under any special agreement, are partners, and as such they are subject to the jurisdiction of equity; but a sale or an assignment by one member of his interest in the lease does not terminate a mining partnership; and a deed of trust by one such partner on his share operates as a sale to the trustee, but it does not constitute the trustee a member of the partnership.

Wetzel *v.* Jones (West Virginia), 84 Southeastern, 951, p. 952, April, 1915.

DISSOLUTION AND TERMINATION OF LIABILITY.

An agreement by and among three lessees of mining property to work and operate a mine under and in pursuance of the lease is sufficient to create a mining partnership in its narrowest and most technical sense; but such a partnership terminates when any one of the lessees ceases to work or aid in the working of the mine, and the other lessees thereafter have no authority to operate the mine in behalf or at the expense of the lessee who has positively withdrawn from the operation of the mine.

Lamont *v.* Reynolds (Colorado), 144 Pacific, 1131, p. 1132, December, 1914.

METHODS OF OPERATING.

MINING COAL NOT INTERSTATE COMMERCE.

The mere act of mining coal is not interstate commerce.

Delaware, etc., R. Co. *v.* Yurkonis, 220 Fed., 429, p. 433, January, 1915.

LABOR ORGANIZATIONS.

RELATION OF CAPITAL AND LABOR.

The industrial development of the world within the last half century has been such as to render it necessary for the courts to take a broader and more comprehensive view than formerly of questions pertaining to the relation that capital sustains to labor.

Mitchell *v.* Hitchman Coal & Coke Co., 214, Fed., 685, p. 698.

LABOR UNIONS LAWFUL.

Some expressions in the earlier English cases indicate that labor unions in England were formerly regarded as unlawful and this rule was applied especially to labor unions formed for the purpose of securing a higher rate of wages, and they were regarded by the English courts as criminal conspiracies; but this rule has never been recognized by the courts of the United States and the rule throughout this

country is that labor may organize for its own protection and to further the interests of the laboring classes, and laborers may "strike" and persuade and induce others to join them. It is only when such a union of laborers resorts to unlawful means to cause injuries to others, to whom they have no relation, contractually or otherwise, the limit permitted by law is passed and they may be restrained.

Mitchell *v.* Hitchman Coal & Coke Co., 214 Fed., 685, p. 697.
Bittner *v.* West Virginia-Pittsburgh Coal Co., 214 Fed., 716, 717.

RELATIVE RIGHTS OF OPERATORS AND MINERS—CONTROVERSIES SETTLED BY COURTS.

So long as capital employs legitimate means for the protection of property rights, it is to be accorded the protection of the law; but this does not mean that capital may, by improper methods, form combinations for the purpose of preventing labor from organizing for mutual protection; and laboring men have the right to use peaceable and lawful methods and unite their forces in order to improve their condition as respects their ability to earn a decent living, give their children moral and intellectual training and secure the enactment of legislation requiring mine owners to adopt such methods as may be necessary to keep their mines in a sanitary condition; and likewise to adopt methods to minimize, as much as possible, the occurrence of the awful catastrophes by which so many human lives have been lost; and when a controversy arises between labor and capital the use of dynamite or any other unlawful methods on the part of the representatives of labor, whereby property and human lives are destroyed, is not to be tolerated by the courts. The relative rights of the parties in all such controversies are entitled to equal consideration and controversies between mine employers and labor unions will be dealt with in the same spirit that actuates the courts in adjusting differences between individuals in questions affecting ordinary transactions; and until it is provided by legislation that labor disputes shall be settled by arbitration the courts must determine all controversies of this character, and as the law now exists, when property or personal rights are involved, the courts alone can furnish adequate relief.

Mitchell *v.* Hitchman Coal & Coke Co., 214 Fed., 685, p. 715.

COAL MINERS—FOREIGNERS PROTECTED—UNITED MINE WORKERS.

A large percentage of the million or more foreigners landing on our shores annually are employed as miners, and while vast numbers of them are unable to read and write or understand the English language, they believe they can secure a more substantial recognition of their rights as members of a labor union; and so long as they are here for the purpose of earning a living and improving their condi-

tion, and at the same time adding to the wealth of the country, it is the duty of the Government to afford them equal protection under the Constitution; and there is nothing in the national constitution of the United Mine Workers of America or in the rules of this organization that binds a member contrary to his wishes or prevents him from exercising his own free will; nor has the union arbitrary power, by violence, intimidation, or otherwise, to compel a miner to become a member of the organization; nor do the rules of the organization control or abrogate or destroy the right of an employer to contract with the members independent of the organization, and this as well as similar organizations may use all lawful methods for the purpose of inducing others to join its order, and, until the contrary appears, it must be assumed that only lawful methods are to be employed for the accomplishment of such purpose.

Mitchell *v.* Hitchman Coal & Coke Co., 214 Fed., 685, pp. 699–702.

UNLAWFUL ACTS OF LABOR UNIONS.

If the United Mine Workers of America in any instance resort to coercion, threats, intimidation, or violence for the purpose of preventing a mine owner from employing nonunion men, such conduct would be unlawful, and the courts would promptly restrain anyone who might be a party to such transaction; and it would always be unlawful for an individual to undertake, by coercion, intimidation, or threats, to prevent a mine owner from exercising his own free will as to the employment of nonunion laborers, or as to any other thing which he might deem necessary to be done in order to protect his property rights.

Mitchell *v.* Hitchman Coal & Coke Co., 214 Fed., 685, p. 703.

PREVENTING OPERATIONS—INJUNCTIONS.

STRIKING MINERS—VIOLATIONS OF INJUNCTION.

It is a violation of an injunction issued to restrain miners from committing any violence against the property and nonunion employees of a mining company where a miners' union and its members continue to threaten assaults, fire shots into the mine inclosure, and throw stones at the employees, and where it was made necessary by reason of the threats and intimidation of the strikers to have the mine guarded by United States marshals and guards were required to accompany employees sent to purchase supplies, and where language was used and conduct continued intended to incite strikers and others to a violation of the restraining order.

United States *v.* Colo., 216 Fed., 654, p. 656.

INTERFERENCE WITH WATER MAINS ENJOINED.

Where there is a statutory right to lay a water pipe line from a watercourse over certain reservations to mining operations on leased premises, and where after opening one mine and laying one pipe line by the lessee it is found impracticable, on account of mining conditions at that point, to successfully and profitably mine, coal, and to perform the covenants of the lease to use the best and most improved methods and to pay the rent and royalties reserved, it accordingly becomes necessary to abandon such operations, locate a new mine and lay a new water line thereto across the reserved lands, and when such new plant and water line have been so located and laid at great expense and used and operated for two years or more without interference by the lessor, equity will enjoin the lessor from thereafter interfering therewith as well as from prosecuting suits for damages for the maintenance of such new water line.

Mary Helen Coal Co. *v.* Hatfield (West Virginia), 83 Southeastern, 292, November, 1914.

MINE OPERATOR LIABLE FOR NUISANCE.

A mine owner operating his coal mine on or near the banks of a creek, and who in pursuit of the business of mining has erected a number of tenant houses, stables, and outhouses, together with slaughter pens where cattle are killed and dressed, and where all the refuse and filth from these various establishments fall into or are washed into the waters of the creek and carried down upon or along the lands of a lower riparian owner, may be liable in damages to such lower riparian owner for maintaining a nuisance.

Stouts Mountain Coal & Coke Co. *v.* Tedder (Alabama), 66 Southern, 619, November, 1914.

MINING LEASES.

LEASES GENERALLY—CONSTRUCTION.

CONSTRUCTION OF NOTE AND MINING LEASE—RATE OF INTEREST.

A note given in part payment for the purchase of mining property may be construed in connection with a mining lease executed at the same time for the purpose of determining the rate of interest on the note.

Boswell *v.* Big Vein Pocahontas Coal Co., 217 Fed., 822, p. 823, November, 1914.

LEASE AS A SALE.

A lease of mines is not in reality a lease in the sense of an agricultural lease, as there is not sowing and reaping, and in the ordinary sense of the term there are no periodical harvests; but what is called

a mineral lease is really, when properly considered, a sale out and out of a portion of the land.

Von Baumbach *v.* Sargent Land Co., 219 Fed., 31, p. 38.

LESSEE LIABLE TO LESSOR FOR COMMISSION FOR OBTAINING LEASE.

The owner of mineral lands is entitled to an accounting and to recover from a person who induced him to execute a mining lease under certain stated terms, on the false and fraudulent representations that the lessee had arranged to transfer and assign the lease to a third person for a stipulated sum, and where the lessee agreed to pay such owner, as an additional inducement to execute the lease, a stipulated part of the sum so to be received by him in consideration of the assignment of the lease, and where the sum stated by the lessee to be received by him for the assignment of the lease, was in fact much less than the actual sum he did subsequently receive for the assignment of the lease.

Expansion Realty Co. *v.* Geren (Missouri Appeals), 170 Southwestern, 928, p. 923, November, 1914.

NATURE AND CONSTRUCTION—RIGHTS OF LESSOR AND LESSEE.

A lease by which the lessee is authorized to explore for, mine, and remove the merchantable iron ore which may be found in or upon the land, and to construct buildings, make excavations, openings, drains, roads, and other improvements thereon, suitable for mining and removing the iron ore, with the privilege of cutting and using timber from the leased premises for carrying on mining operations, with provisions for the payment of the ore mined and giving the lessor a lien upon all ore taken from the mine and upon all improvements as security for the payment of the royalties for ore mined, confers upon the lessee the right to explore for, remove, and transport to market all iron ore found therein, and such additional rights and possession of the control of the premises as are necessary to the proper conduct of the mining operations; but the lessor as owner of the fee retains the right of possession of the surface of the land and can maintain an action against third persons entering thereon without right or authority from him.

Howell *v.* Cuyuna Northern Railway Co. (Minnesota), 149 Northwestern, 942, December, 1914.

TERMINATION OF LEASE—PROOF OF CUSTOM.

Under a mining lease by which the lessee obligated himself to work the mine property as steadily and continuously during the term as the weather and seasons of the year would permit, the lessee can not, in an action by the lessor for rent due, prove a custom among miners

by which lessees under such leases are permitted and have the right to cease working under the lease and remove from the leased premises whenever they choose so to do, as proof of such a custom would impair the lessor's right to recover rent provided for, and further for the reason that the terms of such a lease as to the time of operation are plain and such a lease can not be varied by evidence of usage, as such proof can be shown only when the terms of a lease are obscure or uncertain and the lessee was bound to continue the work as steadily and as continuously from the date of the lease as the weather and seasons of each year would permit during the entire term of the lease.

Northern Light Mining Co. *v.* Blue Goose Mining Co. (California), 143 Pacific, 540, p. 542.

LESSEE'S RIGHT TO CONSTRUCT RAILROAD ON SURFACE.

A lessee under a mining lease authorizing him to construct all buildings, make all excavations, openings, drains, railroads, wagon roads, and other improvements, on the leased premises suitable for the mining and removing of iron ore and granting such other facilities as may be necessary to a successful operation of a mine, has no right to authorize the construction of a railroad upon the leased premises, except for the purpose of aiding in the mining operations and the transportation of ore to market.

Howell *v.* Cuyuna Northern Railway Co. (Minnesota), 149 Northwestern, 942, p. 943, December, 1914.

MINING LEASE—USE OF SURFACE—VALIDITY.

A mining lease for a period of 50 years executed for the purpose of conferring upon the lessee the right to mine iron ore in and from the land described, with permission to erect buildings, put in engines and machinery, build, or authorize to be built, roads, including railroads and switches, for the purposes of transporting supplies to any mine on the property and transporting ore therefrom, and to do all such other things as may be necessary or convenient for the carrying on of iron-ore mining on the premises described, is valid though the surface of the land is used for agricultural purposes, and does not contravene the constitutional provision that "no lease or grant of agricultural lands for longer than 12 years, reserving any rent or surface, shall be valid."

De Grasse *v.* Verona Mining Co. (Michigan), 152 Northwestern, 242, p. 250, April, 1915.

OPTION AGREEMENT TO LEASE—EFFECT AND CONSTRUCTION.

An option agreement giving the leasing party the right to demand and receive a mining lease at any time within one year upon the condition that the lessee would commence within a stated period, and

continue in a manner specified, the exploration of the property until a demand for a lease or abandonment of the option, obligates the optionor to execute and deliver a lease on demand, whether, when such demand was made, ore was discovered or not, on the sole condition that up to the time of such demand the optionee kept the option alive by commencing and continuing the required exploratory work; and the optionor can not be heard to say he was induced to execute the lease upon a misrepresentation of a certain fact, when he was bound by his contract to execute the lease whether the fact alleged to have been represented was true or otherwise.

De Grasse *v.* Verona Mining Co. (Michigan), 152 Northwestern, 242, p. 245, April, 1915.

COVENANT TO EXPLORE AND DEVELOP NOT IMPLIED.

A lease providing that all mines of iron ore on the premises described shall be opened, used, and worked in such manner as is usual and customary in successful mining operations of similar character when conducted by the proprietors themselves, and so as not to do or permit any unnecessary or unusual injury to the same, or inconvenience or hindrance in the subsequent operations of any mine or mines on the leased premises, the lessee not to remove or impair any support, timber, framework, shafts, tramsways necessary or convenient for the use or maintenance of such mines, and providing that the earth and rocks taken from the mine shall be dumped in such places as not to interfere with the convenient working of any mine or mines, does not require the lessee to open, use, or work a mine; but it rather fixes and settles the manner in which the lessee shall open, use, and work a mine, the object being to prevent a waste of ore and unapproved methods of mining, and to compel the lessee to leave the mine on its surrender in a good, safe, workmanlike condition; and the court can not under the statute of Michigan, which provides that no covenant shall be implied in any conveyance of real estate, whether the conveyance contains special covenants or not, create a covenant by implication in the lease and impose on the lessee an obligation to operate the mine or make thorough exploration to determine whether the leased premises contain ore or not.

De Grasse *v.* Verona Mining Co. (Michigan), 152 Northwestern, 242, p. 247, April, 1915.

RIGHT TO REMOVE ORE MINED.

A lease executed under an option providing for a lease for mining purposes and removing iron ore from the described premises, and which confers upon the lessee the right to mine and provides for the payment of royalty on all ore mined and removed from the premises, and provides that the lessee shall pay a stipulated sum each year if

he fails to mine and remove less than a stated amount each year, clearly discloses an intention on the part of both parties to include in the lease the right of removal of the iron ore mined.

De Grasse *v.* Verona Mining Co. (Michigan), 152 Northwestern, 242, p. 249, April, 1915.

CONSTRUCTION—DIAMOND LEASE—FRAUD.

A mine owner and lessor of a diamond mine can not recover the possession of diamonds from the lessee, mined by him from the leased mine, on the ground that the lease was procured for the fraudulent purpose of discrediting the mine and to keep the lessor ignorant as to the value of the large diamonds discovered, to stifle the business of mining, to depress the value of the mine, and to buy it for a minimum price, unless he proves by a fair preponderance of evidence the fraudulent acts alleged by him and that the lease was procured for the fraudulent purposes alleged and that the diamonds were in fact mined with the false and fraudulent intent as averred.

Mauney *v.* Millar (Arkansas), 175 Southwestern, 402, March, 1915.

ROYALTIES.

Royalty is a certain percentage or proportion specifically stated or on a graduated scale according to the value of the ore, based on either the net proceeds, smelter returns, mill returns, or returns evidenced by the certificate of the United States assay office, or otherwise, as the parties agree upon.

Saulsberry *v.* Saulsberry (Kentucky), 172 Southwestern, 932, February, 1915.

RECOVERY OF RENT OR ROYALTY.

In an action on a lease by which the lessee agreed to work certain mines in mine fashion and pay a certain stipulated rent or royalties, the amount depending on the amount of mineral mined, the royalties to be paid annually in gold or precious minerals extracted from the premises, no allegation of a demand for the payment of the rent is essential though no particular day was designated for payment; but the precise time and whether made in one or more installments was left to the discretion of the lessee, but in any event the entire amount of annual rent would be due at the end of the year, and if not paid an action would then lie for its recovery, whether the payment was to be made in gold dust, bullion, mineral or money.

Northern Light Mining Co. *v.* Blue Goose Mining Co. (California), 143 Pacific, 540, p. 542.

RIGHT TO ROYALTIES FROM SUBLEASE—TRANSFER BY BANKRUPT.

An owner of an undivided interest in a mining claim leased his co-owner's interest and agreed by the terms of the lease to work the entire claim, and thereafter sublet the claim to another reserving to

himself a percentage of the gross output, and then conveyed his undivided interest in the claim to his wife and subsequently became a bankrupt. In a contest between the trustee of the estate of the bankrupt and the bankrupt's wife, the latter was not permitted to retain all the royalties reserved in such sublease as against the trustee in bankruptcy.

Stroecker *v.* Patterson, 220 Fed., 21, February, 1915.

FORFEITURE—EFFECT ON ROYALTIES NOT DUE.

A forfeiture of a mining lease by the lessor and a reentry by him between rental periods, releases the lessee from liability for all rents not fully accrued at the time of such forfeiture and reentry.

G. W. Young's Mining Co. *v.* Courtney, 219 Fed., 868, p. 871.

ARBITRARY FORFEITURE—EQUITABLE RELIEF DENIED.

A lessor of a mining lease who has arbitrarily declared a forfeiture of the lease and who has reentered and taken possession of the leased premises, will not be granted equitable relief and can not have an equitable lien declared upon the lessee's personal property for royalty due, where the lessee had not produced sufficient ore in the current year to entitle him to a setoff for the excess of royalties paid; but where having a contingent right thereto, it would have been available but for the arbitrary forfeiture of the lease.

G. W. Young's Mining Co., 219 Fed., 868, p. 872.

FORFEITURE—BREACH OF IMPLIED COVENANT.

Where the right of forfeiture in a mining lease is confined to the failure of the lessee respecting the covenants and conditions which are expressed in the lease, and does not arise upon the nonobservance of an implied covenant or condition, the lessor can not claim the right to forfeit the lease because of the failure of the lessee to perform an implied covenant.

De Grasse *v.* Verona Mining Co. (Michigan), 152 Northwestern, 242, p. 249, April, 1915.

AVOIDANCE FOR FRAUD—KNOWLEDGE OF FACTS.

To entitle a lessor of a mining lease to cancel and avoid the same on the ground that he was induced to enter into it upon the faith of unwarranted representations, it is essential that he should be ignorant of the matters represented; and if before he executed the lease he had knowledge of the truth and knew that the statements made by the lessee and incorporated in the lease itself were not true, he can not claim to have been deceived; and where the mining lease recites that the lessee had explored the premises and represents that there

exists thereon iron ore in paying quantities, if the lessor was well informed of the results of the exploratory work, and knew that iron ore in quantity and quality sufficient to warrant opening up and mining the same had not been discovered, he can not claim to have been defrauded by the representation.

De Grasse v. Verona Mining Co. (Michigan), 152 Northwestern, 242, p. 244, April, 1915.

COAL LEASES.

PRACTICAL CONSTRUCTION.

A court will not construe a coal lease, though somewhat ambiguous, where the parties themselves continued payment of royalties for a period of three years after the death of the lessor, and thereby placed their own construction upon the lease with reference to the time and method of making payment of royalties.

Prudence Coal Co. v. Perkins, 217 Fed., 569, p. 574.

CONTRACT AS A SALE AND NOT A LEASE—FORFEITURE AND CANCELLATION.

A contract, denominated a lease, by which the lessee named was given the exclusive right to mine and remove within a stated period all of a certain vein of coal underlying certain tracts of land, providing that the lease was in gross and not by acreage and reciting that it sold outright and transferred to the lessee a mining plant and certain property and machinery used in connection therewith, for a stipulated consideration, payable in different amounts at stated dates, and providing also for the payment of a "long-ton royalty of 15 cents to be paid for the coal mined and removed," and providing further that the royalty should for the first three years amount each year to a stipulated sum payable in quarterly installments and that thereafter it should amount to another stated sum payable in quarterly installments, conditioned that the lessor could forfeit the lease on failure of the lessee to keep and perform its terms and conditions, is in fact a mining lease and not a sale of either land or coal by the acre, nor of any fixed quantity or number of tons, but a right in gross to mine and remove all the coal in the particular vein described; and the stated sum, aside from the royalty named and so much of the sum named as a consideration as was in excess of the value of the mining plant and the property connected therewith, must be regarded as a bonus for the privilege of mining the coal on which the royalty was to be paid; and such a lease may at the instance of the lessor be canceled as a cloud on the title on failure of the lessee to operate.

Browning v. Boswell, 215 Fed., 826, p. 834.
See Boswell v. Big Vein Pocahontas Coal Co., 217 Fed., 822.

LEASE OF SEPARATE TRACTS—TIME LIMIT TO EACH—ENTIRE CONTRACT.

A coal lease by which the lessor grants to the lessee the right to mine the coal from under several separate tracts and parcels of land at stipulated royalties to be paid at certain stated times, the period of duration of the lease being different as to different tracts of the land leased, the lease containing an obligation on the part of the lessee to mine the coal from under the tract where the time limit is the least, at the earliest moment practicable, so that the coal may all be mined before the time limit expires, is an entire contract, and indicates no purpose or intent to give the lessee a right to avail himself of some of its provisions and at the same time abandon other portions of the lease; and if the lessee desires to avail himself of the provisions of the lease he must proceed without unreasonable delay to mine the coal from under the land where the time limit is the least, and if possible to have the coal mined from such land before the time limit expires; and on the failure of the lessee to proceed under the lease and begin to mine coal until the time limit of some of the tracts had expired, this works a forfeiture of the entire lease at the option of the lessor.

Ray Coal Mining Co. *v.* Ross (Iowa), 151 Northwestern, 63, February, 1915.

SPECIFIC PERFORMANCE NOT DECREED.

A court of equity has no power to decree specific performance of a mining lease or to compel the lessee to mine and remove the coal, nor can the court divide up and abate in part an indivisible right to buy the mine or remove the coal, but the only remedy for the defrauded party is to have the lease canceled.

Browning *v.* Boswell, 215 Fed., 826, p. 836.

IMPLIED SURRENDER.

An implied surrender of a coal lease does not arise from mere notice of intention on the part of the lessee not to pay further rentals and to give up the lease, and his failure to make such payments; but where there is an abandonment of actual operations under a lease, and the lessor reenters or executes another lease, these circumstances establish an implied surrender.

Laing *v.* Price (West Virginia), 83 Southeastern, 497, p. 499, October, 1914.

COVENANT TO PAY MINIMUM RENTAL—RELIEF.

A covenant in a coal lease to pay fixed or minimum rentals is ordinarily absolute and enforceable; and while such covenants are sometimes relieved against as in case of leasing unknown and undeveloped territory, or where there are terms of express or implied warranty,

or where the parties are mutually mistaken as to the existence of mineral, yet a lessee can not avoid his absolute promise to pay a minimum rental where there has been no real effort on his part to determine whether any of the veins of coal are susceptible of advantageous mining, where no entry has ever been driven, no shaft sunk on the premises, or ore taken from the land, and where it appears that coal seams actually exist in the leased premises at a thickness of at least 3 feet and a very considerable vein of coal runs through the leased premises.

Laing v. Price (West Virginia), 83 Southeastern, 497, p. 499, October, 1914.

CONSTRUCTION—PAYMENT OF ROYALTIES.

An agreement in a coal lease to the effect that the lessee may continue to mine the coal from the leased premises, paying the lessor at the rate of 25 cents per ton for his portion of the coal so mined, agreeing to other established usage and practice, does not authorize the lessee to pay royalty only on coal that passed over a $\frac{5}{8}$-inch mesh and to appropriate all the smaller sizes that passed through such mesh without paying any royalty thereon.

Gerard Trust Co. v. Delaware & Hudson Co. (Pennsylvania), 92 Atlantic, 129, p. 130, July, 1914.

ACTION FOR ROYALTIES—DEFENSE.

In an action by a lessor against the lessee under a coal lease for royalties for coal mined under such lease, it is no defense to the action for the lessee to aver that at the time of the execution of the lease the lessor did not have title to the leased premises, in the absence of an allegation of fraud, accident, or mistake in the execution of the lease, where it is not averred that the lessee had been evicted from or had surrendered the possession of the tract, or that the lessor warranted the title.

Lazarus v. Lehigh & Wilkes-Barre Coal Co. (Pennsylvania), 92 Atlantic, 121, p. 123, July, 1914.

LESSEE MAY COMPEL ADJUSTMENT OF ROYALTIES.

Under a coal lease executed by an agent for heirs where there is a subsequent controversy as to the persons who were entitled to royalties under the lease, the lessee may bring a suit against all the parties claiming any interest in the royalties under the lease, and tender or pay the amount due from him under the lease and compel the other parties to determine their rights to the royalties thus paid or tendered.

Wilmer v. Philadelphia, etc., Coal & Iron Co. (Maryland), 93 Atlantic, 157, January, 1915.

ROYALTY—RENT AND ROYALTY SYNONYMOUS.

The word "royalty" as employed in a coal-mining lease means a share of the profit reserved by the owner for permitting the removal of the coal and is in the nature of rent; and while it may be conceded that royalty is a mere property rent where rental is based upon the quantity of coal or other mineral that is or may be taken from the mine, yet it can not be doubted that the terms "rent" and "royalty" as the result of usage and custom are often used interchangeably, and accordingly mining leases are made every day where the term "rent" is employed, even though it may as a matter of fact assume the form of royalties.

Saulsberry *v.* Saulsberry (Kentucky), 172 Southwestern, 932, p. 933, February, 1915.

RENTS.

Rent is defined as something given by way of compensation to a lessor for the right to make use of land demised, and when applied to coal lands and leases it means the profit or return reserved, payable periodically, but not necessarily immediately, if it issues from period to period, during the whole continuance of the grantee's estate, whether from year to year, or from month to month.

Saulsberry *v.* Saulsberry (Kentucky), 172 Southwestern, 932, p. 933, February, 1915.

AGREEMENT TO MINE COAL AND PAY ROYALTY—EFFECT AS A LEASE.

An agreement to the effect that the second party may continue to mine the coal from the land held in common by the first and second parties, paying the first party at the rate of 25 cents per ton for his portion of the coal mined, does not constitute a sale of the coal in place, or a conditional sale of the coal in place, but does create the relation of landlord and tenant.

Gerard Trust Co. *v.* Delaware & Hudson Co. (Pennsylvania), 92 Atlantic, 129, p. 130, July, 1914.

OIL AND GAS LEASES.

CONSTRUCTION AND CONSIDERATION—CONTRACT FOR OPTION.

An oil lease by which the lessee has the right to bore for oil or pay a small quarterly rental, or to surrender the lease at any time upon the payment of a mere nominal sum to the lessor, is based on a mere nominal consideration and does not pass an interest in the land and is a mere contract for an option by which the lessee may acquire an interest in the land.

Owens *v.* Corsicana Petroleum Co. (Texas Appeals), 169 Southwestern, 192, p. 194, June, 1914.

CONSTRUCTION—RIGHTS AND DUTIES OF LESSEE.

A different rule is applied to oil leases from that applied to ordinary leases, and leases of land for the purpose of being prospected and developed for oil are construed most favorable for the lessor; and a lessee under such a lease, in order to maintain his rights, must begin within a reasonable time the performance of obligations imposed upon him by the lease and continue in the performance thereof with reasonable diligence, as the discovery and production of oil is a condition precedent to the continuance or vesting any estate in the demised premises; and where the only real consideration is prospective royalty to come from exploration and development, a failure on the part of the lessee to develop the premises renders the agreement a mere nudum pactum and works a forfeiture of the lease. Such a lease may be forfeited at the option of the lessor where the lessee has not only failed to develop the premises demised, but especially so where he has leased the adjoining premises and has drilled wells near the line of the leased premises thereby robbing the lessor of his due share of the underlying oil.

Owens *v.* Corsicana Petroleum Co. (Texas Appeals), 169 Southwestern, 192, p. 198, June, 1914.

JURISDICTION OF COURT TO MODIFY OR CANCEL.

A county court of Oklahoma is without jurisdiction to entertain a petition of a lessor in an oil and gas lease to modify and cancel the same, where such lease was executed by the former guardian of the petitioner on the order of a proper court and approved by the Secretary of the Interior.

Ozark Oil Co. *v.* Berryhill (Oklahoma), 143 Pacific, 173.

INDIAN LEASE—CONDITIONS PRECEDENT—APPROVAL BY SECRETARY.

Provisions in an order of court authorizing a guardian of an Indian to join with the latter in the execution of an oil and gas mining lease on the allotment of such Indian, and providing that the lease should be subject to the approval of the Secretary of the Interior and should be executed in accordance with the rules and regulations prescribed by him, and directing the guardian to make a full report when the order had been complied with, and directing that a stipulated bonus should be placed in escrow in a certain designated bank until the Secretary had approved the lease, can not be considered as idle and useless provisions, but they must be taken as having a fixed and set purpose understood by the court and the parties; and the provision requiring the approval of the Secretary of the Interior was a condition precedent to be complied with in order to complete the execution of the lease contract, and not having been complied with the

contract was not completed and no estate vested in the lessee under the lease.

Wellsville Oil Co. *v.* Miller (Oklahoma), 145 Pacific, 344, p. 348, December, 1914.

RIGHTS TO USE OF SURFACE—LOCATION OF WELLS.

The lessee and lessor under an oil and gas lease are both in possession of the surface and each, in the exercise of his right therein and thereon, is in duty bound to have due regard for the rights of the other; and the lessee in exercising his rights under such a lease owes the duty to the lessor to not unnecessarily, carelessly, or wantonly injure him in the proper use of the surface; and in choosing between two locations for drilling a well equally available to him, the lessee is bound to choose the one to do the least injury to the lessor, and he is not at liberty to choose locations for the drilling of wells in utter disregard of the rights of the lessor. Likewise the lessor in the use of the surface for any available purposes is in duty bound to exercise reasonable care not to interfere with, injure, or annoy the lessee in drilling and operating his oil wells, and under such circumstances each is bound to use his own so as not to injure the rights of the other.

Gillespie *v.* American Zinc & Chemical Co. (Pennsylvania), 93 Atlantic, 272, p. 273, January, 1914.

PRIOR RECORDED LEASE AS NOTICE.

A lessee of oil lands is to be charged with notice of a prior recorded lease and proof of prior oral negotiations as to what lands the lessor was leasing to the lessee or how much land the lessee thought he was leasing is immaterial, as the rights of the parties are to be determined by the lease itself.

Loeb *v.* Conley (Kentucky), 169 Southwestern, 575, p. 578, October, 1914.

EFFECT OF RECORD.

The fact that an oil lease is void for want of mutuality does not affect its recordability and when recorded it must operate as notice to a subsequent lessee.

Loeb *v.* Conley (Kentucky), 169 Southwestern, 575, p. 578, October, 1914.

LEASE VALID WITHOUT RECORDING—TITLE TO OIL AND GAS.

A lease demising and granting the oil and gas and other minerals in a tract of land for a term of years and for as much longer time as oil and gas and other minerals are found in paying quantities thereon, the lessee to pay the lessor a stipulated royalty on all oil and gas produced, the lease binding the lessee to commence a well within 60 days, but containing no obligation requiring the lessee to prosecute

mining operations further, does not transfer the title to the oil and gas or to other minerals, and the lease was not required to be recorded or listed for taxation in order to be valid.

Finch *v.* Beyer (Kansas), 146 Pacific, 1141, p. 1142, March, 1915.

ROYALTIES.

The word "royalty" as used in an oil and gas lease, generally refers to a share of the product or profit reserved by the owner for permitting another to use the property.

Saulsberry *v.* Saulsberry (Kentucky), 172 Southwestern, 932, p. 933, February, 1915.

DEVELOPMENT IMPLIED—DILIGENCE REQUIRED IN OPERATION.

A lessee under a lease for the production of oil and gas, containing the usual terms and conditions, must, if either oil or gas is found in paying quantities, exercise due and reasonable diligence in prosecuting operations under the lease for the mutual benefit of himself and his lessor; and if he unreasonably fails or refuses so to do, damages therefor may be recovered against him in an appropriate action at law. But the judgment of the lessee as to the diligence with which and extent to which wells should be drilled under the lease, upon discovery of either oil or gas in paying quantities, will control, if exercised in good faith and not unreasonably or arbitrarily to promote his own peculiar benefit to the manifest prejudice of the lessor; and both lessee and lessor are bound by that degree of diligence which, surrounding circumstances and conditions being considered, would reasonably be expected of operators of ordinary prudence, experience, and engaged in the same business, having due regard for the interests and advantage of themselves and their lessors.

Grass *v.* Big Creek Development Co. (West Virginia), 84 Southeastern, 750, p. 752, March, 1915.

IMPLIED COVENANT TO DEVELOP.

In oil and gas leases generally there is an implied covenant on the part of the lessee to begin operations within a reasonable time, and on failure to do so he will be presumed to have abandoned his rights, and a court of equity will at the suit of the lessor cancel the lease as constituting a cloud upon the title.

Horse Creek Coal Land Co. *v.* Trees (West Virginia), 84 Southeastern, 376, p. 378, February, 1915.

It is an implied condition or covenant of every lease of land for the production of oil therefrom that when the existence of oil in paying quantities is made apparent, the lessee shall put down as many wells as may be reasonably necessary to secure the oil for the common advantage of both the lessor and the lessee.

Highfield Co. *v.* Kirk (Pennsylvania), 93 Atlantic, 815, p. 817, January, 1915.

EXCESSIVE OPERATION AS A BREACH.

A plain and substantial disregard of the duty of an oil lessee to operate the lease in such a manner as would be reasonably expected of operators of ordinary prudence, having regard to the interests of both the lessor and lessee, would be a breach of the lease; and the obligation to develop premises held under an oil lease might be a violation as much by a too strenuous as by a too dilatory operation of the lease, and such a method of operating an oil lease can not be made the basis of an equitable right to compel the execution of a new lease.

Wellsville Oil Co. *v.* Miller (Oklahoma), 145 Pacific, 344, p. 347, December, 1914.

UNILATERAL LEASE—RIGHTS OF LESSEE—FAILURE TO DEVELOP.

An oil lease based upon a mere nominal consideration, leaving it optional with the lessee to bore for oil or to pay a small quarterly rental, is unilateral and void, unless the lessor accepts the rents at the expiration of each quarterly period and thereby agrees to continue the option for three months longer; and the lessee under such a lease has no right that can be enforced, in the absence of a showing that he had in good faith attempted to explore for oil. Such a lease can not be construed so as to enable a lessee to hold it merely for speculative purposes without doing what he stipulated to do and what was clearly in contemplation of the lessor when he executed the lease.

Owens *v.* Corsicana Petroleum Co. (Texas App.), 169 Southwestern, 192, p. 194. June, 1914.

LIABILITY OF LESSEE FOR GAS WELLS.

An oil and gas lease providing that the lessee shall deliver to the credit of the lessor free of cost in a pipe line, one-eighth part of all the oil produced and saved from the leased premises and pay $300 per year for the gas from each and every gas well drilled on the premises, must be interpreted in the light of all the facts and circumstances surrounding the parties, their relation to each other, the objects and purposes of entering into the contract, and the term "gas well" as used in the contract must be taken to mean a gas well, which, considering its location with reference to any market for gas, its capacity as a gas producer, that it can be profitably operated as such, and not a well producing oil in large quantities and some gas and operated by the lessee for many years as an oil well, and without demand for gas rental by the lessor; and the fact that some gas is found in a well and is run from the casing head into a gas line from wells on an adjoining lease by the lessee, and the gas from all utilized in operating the wells on both properties, according to a custom

prevailing among oil operators, does not impose a liability on the lessee.

Prichard *v.* Freeland Oil Co. (West Virginia), 84 Southeastern, 945, p. 946, April, 1915.
Locke *v.* Russell (West Virginia), 84 Southeastern, 948, p. 950, April, 1915.

PRODUCING WELLS—COLLECTING GASOLINE AS EVIDENCE OF GAS.

The fact that the lessee of an oil and gas lease who had drilled and was operating oil wells, installed and connected vacuum pumps in connection with such wells for the purpose of increasing the production thereof, and the further fact that the lessee successfully utilized what was called or termed "vapor," which was emitted from the wells at the casing head, and by process of distillation and compression converted the escaping substance into gasoline for the mutual advantage and benefit of the lessee and lessor, did not thereby render the lessee liable for the annual rental of gas wells, under the terms of the lease, as the mere collecting of the vapor or volatile substance and the manufacture of gasoline therefrom was no indication of proof of gas in the wells, and did not bring them within the terms of the lease as producing gas wells.

Locke *v.* Russell (West Virginia), 84 Southeastern, 948, p. 949, April, 1915.

DAMAGES FOR FAILURE TO DEVELOP.

A lessor of oil lands for the production of oil and gas in an action against the lessee for failure to properly develop the leased premises is entitled only to such damages as he sustained by any failure on the part of the lessee to exercise an honest judgment in proceeding with the necessary explorations on the leased lands and the extraction of oil therefrom, taking into consideration the subject matter of the lease, the character of the mineral products, the nature of the oil-bearing sand, whether dense or soft and porous, developments on contiguous lands, whether by the lessee or different operators, the cost of drilling, proximity to market, and facilities for marketing, current prices, whether high or low, location of lands, and such other conditions attendant on the operations as may explain the necessity for prompt, or excuse for delayed, action in prosecuting such development. But in such case the lessor assumes the burden of showing, and by clear and convincing proof must, to avail him, show by witnesses having experience, skill, and engaged in similar operations that the lessee, having due regard for the advantage and profit of himself and the lessor, has not, surrounding circumstances and conditions considered, exercised ordinary diligence in conducting such operations.

Grass *v.* Big Creek Development Co. (West Virginia), 84 Southeastern, 750, p. 753, March, 1915.

FAILURE TO DEVELOP—FORFEITURE—ESTOPPEL.

A lessor of oil lands for the purpose of having the same developed by the lessee may, by permitting the lessee to expend large sums of money in drilling a well, waive his right to declare a forfeiture because of the failure of the lessee to develop the land within the time specified in the lease.

Owens *v.* Corsicana Petroleum Co. (Texas Appeals), 169 Southwestern, 192, p. 195, June, 1914.

TERMINATION ON FAILURE TO DRILL WELL OF AMOUNT.

Where parties by an oil lease and contract fix and agree upon a limit of time within which the oil company may explore for oil, and stipulate that by which the bringing in of a well of stated daily production gives the right to continue upon the land for an indefinite time, the parties must be bound by the terms of their contract, and the courts are not at liberty to extend or enlarge or vary the terms because of misfortune or accident of the oil company not provided for in the contract, which, without any fault on the part of the lessor, prevented the lessee, the oil company, from successfully bringing in a well of the stipulated daily production; and under such a contract when the time limit expires, and the oil company has failed to produce oil in the quantities stipulated, the lessor has the right to declare the lease at an end.

McLean *v.* Kishi (Texas Appeals), 173 Southwestern, 502, p. 503, January, 1915.

ABANDONMENT.

The failure of the lessee of an oil and gas lease to drill more than one well upon the premises in a period of 16 years, together with the declared intention of the lessee that the quantity of oil in the premises and the cost of drilling another well would not justify further development, shows conclusively an abandonment of the premises, except sufficient of the surface to operate the one well.

Highfield Co. *v.* Kirk (Pennsylvania), 93 Atlantic, 815, p. 816, January, 1915.

FORFEITURE AND CANCELLATION—EQUITY WILL NOT AID IN FORFEITURE.

A lessor of lands for oil can not invoke the aid of a court of equity to declare and enforce a forfeiture of the lease for a failure on the part of the lessee to comply with its terms and to produce oil in paying quantities, in the absence of an allegation or evidence of a fraudulent refusal to comply with the terms of the lease, or to respond to repeated and urgent demands for additional expenditures of money in continuous development, where the lessee has paid

or tendered the lessor his share of the proceeds arising from the productions of oil and gas resulting from the lessee's activity in producing the same, and where no oil has been wasted and where it appears that the lessee has drilled four wells at considerable cost, from three of which there has been no adequate return from his investment and there will be none from two wells, and where the leased lands were located in a new, untried, and undeveloped oil and gas territory, many miles from a readily available market for the products, and where by an enforced forfeiture gross injustice would result to the lessee.

Horse Creek Coal Land Co. *v.* Trees (West Virginia), 84 Southeastern, 376, p. 377, February, 1915.

TERMINATION.

When it is provided in an oil lease that it is terminable at the will of one of the parties it then becomes terminable at the will of either of the parties.

Owens *v.* Corsicana Petroleum Co. (Texas Appeals), 169 Southwestern, 192, p. 197, June, 1914.

MINING PROPERTIES.

TAXATION.

LEGISLATIVE DESIGNATION OF PROPERTY.

The legislature of a State must determine the questions of State necessity, discretion, or policy involved in ordering taxation, and decides when, how, and for what public purposes taxes shall be levied and collected, and selects the subjects of such taxation; and the statute for the levying of taxes should specifically or otherwise enumerate the kinds of property to be taxed, and the rule as applied to the taxation of oil and gas leases and oil-mining property.

Indian Territory Illuminating Oil Co., In re (Oklahoma), 142 Pacific, 997, p. 999.

INTERESTS IN REAL PROPERTY NOT SEVERED FOR TAXATION.

The Legislature of Oklahoma has not selected oil and gas leases, as such, as subjects of taxation, nor has it provided for the severance of the various interests which may be held in real property for purposes of taxation.

Indian Territory Illuminating Oil Co., In re (Oklahoma), 142 Pacific, 997, p. 999.

LIABILITY OF LESSOR OR LESSEE FOR TAXES.

Where the statute does not provide for the severance, for the purposes of taxation, and an oil and gas lease is silent upon the subject, the obligation to pay taxes upon the leased premises devolves upon

the lessor; but the lessee's assumption of the payment of taxes and assessments does not relieve the lessor from his liability nor does it enable the taxing authorities to secure a personal judgment against the lessee.

Indian Territory Illuminating Oil Co., In re (Oklahoma), 142 Pacific, 997, p. 1001.

LEASEHOLD NOT "REAL PROPERTY" WITHIN TAXING LAWS.

Under sections 7304 and 7307, Revised Laws of Oklahoma (1910), real property, which for the purpose of taxation means land and buildings, structures, and improvements thereon, and all rights and privileges thereto belonging, as well as all mines, minerals, quarries, and trees on or under the same, must be listed and assessed in the name of the owner of the land. A leasehold under an oil and gas lease, being a chattel real, is personal property, and does not fall within the designation of "real property."

Indian Territory Illuminating Oil Co., In re (Oklahoma), 142 Pacific, 997, p. 999.

OIL AND GAS LANDS AND LEASES.

Oil and gas, while lying in the strata or earth from which they are produced, constitute a sort of subterranean faera natura which, if taxed at all, must be taxed as real property to the owner of the land under which, for the time being, they may lie, and can not be taxed against the owner who has a mere lease or license to go upon the premises, search for, and if found, take them away; and this is the most scientific method for imposing taxation upon this class of property, as to undertake to tax an oil and gas lease is to undertake to impose a tax upon the illimitable vista of hope. Many instances are known where lessees have paid thousands of dollars bonus for a lease and have discovered no oil, and other instances are known where leases have cost comparatively nothing and oil has been found in enormous quantities. Whether oil is under any particular tract of land is beyond the ken of man until a well has been drilled, and even then no one can foresee how long a well will last or what its production will be. Under the system of taxation devised by the Legislature of Oklahoma, the wealth produced by the oil industry, the production of oil, the capital invested in its production, the oil on hand, and the oil in place, are taxed, and there is no justification in the law for any additional exactions.

Indian Territory Illuminating Oil Co., In re (Oklahoma), 142 Pacific, 997, p. 1001.

MINING CLAIMS SUBJECT TO TAXATION.

The Montana statute for the taxation of mining property is not an exemption provision, but a revenue measure apportioning to the owners of mining claims what the legislature deemed to be their just

proportion of the public burden, and before the additional burden can be imposed, the taxing authorities must ascertain that the conditions authorizing its imposition in fact exist.

Barnard Realty Co. *v.* City of Butte (Montana), 145 Pacific, 946, p. 948, January, 1915.

MINING CLAIM WITHIN CITY LIMITS.

The fact that mining property is located within the limits of a city and has been platted for the purpose of putting the lots upon the market for sale and selling them as there was demand and was thereby given an independent value for town-site purposes, as other lands within the city limits, is not sufficient to make it subject to special taxation under the constitution of Montana in the absence of proof of an intent on the part of the owner to use it for a purpose other than mining.

Barnard Realty Co. *v.* City of Butte (Montana), 145 Pacific, 946, p. 948, January, 1915.

INDEPENDENT USE OF MINING CLAIM.

The constitution of Montana, providing that mines, after purchase from the United States, shall be taxed at the price paid the United States therefor, unless the surface ground is used for other mining purposes and has a separate and independent value, in which case it shall be taxed at its value for such other purposes, intended to bring into the class of taxable property mines and mining claims and to provide a method by which the owners of them might be compelled to bear their equitable portion of the expenses of the government. So long as a mining claim is used and held exclusively for mining purposes, the owner is not required to bear any other burden. When the property has, by reason of its location, acquired a value for some independent use and is devoted by the owner to such use, it then becomes at once subject to taxation at that value, to be ascertained by the assessing officer. By devoting it to this new use the owner thereby creates an estate which, in the eye of the law, is regarded as independent of the original estate and is subject to taxation.

Barnard Realty Co. *v.* City of Butte (Montana), 145 Pacific, 946, p. 947, January, 1915.

COAL MINED FROM INDIAN LANDS—PRIVILEGE TAX.

The statute of Oklahoma providing for the levying and collection of a gross revenue tax from persons and corporations engaged in mining and requiring such persons to file with the State auditor a statement showing the amount and kind of mineral mined and requiring the payment thereon of a gross revenue tax, does not authorize the State of Oklahoma to levy such gross revenue tax on coal mined from the lands of the Choctaw and Chickasaw Indians by

a lessee of such lands, as the agreement with the Indians imposes upon the United States a definite duty in respect to opening and operating the coal mines upon their lands, and the agency selected by the United States for that purpose can not be subjected to an occupation or privilege tax by the State of Oklahoma.

Choctaw & Gulf Railroad Co. *v.* Harrison, 235 U. S., 292, p. 298.

COAL MINED FROM INDIAN LANDS—GROSS REVENUE TAX.

The agreement between the United States and the Choctaw and Chickasaw Indians, ratified and confirmed by Congress June 28, 1898 (30 Stat., 495, p. 510), providing that their coal should remain common property of the members of the tribe and that revenue derived therefrom should be used in the education of the Indian children, and that the mines thereon should be under the supervision and control of two trustees appointed by the President, and that all such mines should be operated and royalties of 15 cents per ton paid into the Treasury of the United States, giving the Secretary of the Interior power to select agencies for carrying out the agreement and authorizes a lease to a railroad company of the Indian coal lands, and coal mined from mines belonging to such Indians is not subject to a gross revenue tax.

Choctaw & Gulf Railroad Co. *v.* Harrison, 235 U. S., 292, p. 296.

TAXATION OF OIL IN TRANSIT.

The statute of Tennessee providing that persons having oil depots, storage tanks, or warehouses, for the purpose of selling, delivering, or distributing oil, and using a railroad car or railroad depots for such purpose, shall pay a privilege tax on such oil, does not authorize the imposition of such privilege tax on an oil company shipping a tank car of oil from its refinery in one State and a carload of barrels from another State to itself in a third and different State for the purpose of filling orders taken by its salesmen, and where the barrels were filled from such tank car without storage of the oil in a depot or warehouse where in some instances a few barrels of oil were temporarily left for customers at the side of the railroad track.

Western Oil Refining Co. *v.* Dalton (Tennessee), 174 Southwestern, 1138, p. 1139, April, 1915.

ROYALTIES NOT INCOME.

A mining lease granting to the lessee the absolute and exclusive right to take out and have all the ore in the land and to remove it at any time within 25 to 50 years, a time sufficient to enable him to remove all the ore, being equivalent to an unlimited time, the lessee agreeing to pay yearly fixed amounts per ton for all the ore taken, and to pay a stipulated minimum amount annually whether ore was

mined or not, is in effect a grant of a part of the corpus of the property, and the lessees become the owners of the ore and the lessors the owners of the claims and of the right to collect the amounts the lessees covenanted to pay for the ores and the transactions are in effect sales of the ore for covenants to pay the purchase price thereof and such a lease is in reality a sale of the ore and the royalties reserved are in fact the purchase price of the ore; and the amounts paid under the name of royalties for the ore taken can not be called or classed as income, but must be regarded as parts of the capital of the corporation, as the lease merely changed the form of the property of the lessor from the ores to the royalties and claims to the purchase price of such ore, which the lessee covenanted to and did pay under the name of royalties, and such sums are not subject to the United States corporation tax act (36 Stat., 111).

Von Baumbach *v.* Sargent Land Co., 219 Fed., 31, p. 37.

STATE BOARD'S POWER TO REASSESS.

The Legislature of Oklahoma may provide for a reassessment of property of public-service corporations which has been assessed at less than its fair value, but in the absence of any such provision the State board of equalization has no power to do so; and where the State board has assessed property of all public-service corporations and has equalized the various county assessments and computed the amount of the State levy and caused the same to be certified to the several county clerks, it is then without jurisdiction or authority to reconvene and reassess such property, and prohibition will lie and is the proper remedy to restrain the State board from so doing.

Prairie Oil & Gas Co. *v.* Cruce (Oklahoma), 147 Pacific, 152, March, 1915.

TRESPASS.

PROPERTY DESTROYED BY BURNING OIL.

An oil-operating company that stored its oil in tanks and thereafter negligently allowed live crude petroleum oil to escape from its tanks and premises, flow down through a ditch and to, against, and around a dredge, the property of another, well knowing that such oil was liable to be ignited and damage or destroy such property, and negligently failed to warn the owner of the property or his servants in charge of the dredge that the oil from the tanks had escaped, and that it was live and inflammable oil, different from dead oil that had previously flowed down and around such dredge, and where the oil was accidentally ignited by fire from the dredge and the dredge was destroyed, is liable for damages for the loss of the dredge by fire, though the oil was accidentally ignited by fire from the dredge, as the negligent act of the corporation in permitting the oil to escape and

flow down off of its premises around the dredge, must be regarded as the proximate cause; and the accidental firing of the oil is not an independent agency that could not reasonably have been anticipated, but it was an occurrence that should have been reasonably expected in the course of events and according to common experience in handling such oil and therefore it did not break the chain of causation extending from the original wrongful act and negligence of the oil operator.

Rock Oil Co. *v.* Brumbaugh (Indiana Appeals), 108 Northeastern, 260, p. 263, March, 1915.
See Commercial Union Assurance Co. *v.* Gulf Refining Co. (Texas Civil Appeals), 174 Southwestern, 874, March, 1915.

CONVERSION—MEASURE OF DAMAGES.

In an action for trespass upon metalliferous veins, where the trespasser has mined, milled, and sold the finished or enhanced product, the conversion takes place when he applies the proceeds to his own use, and the measure of damages is the enhanced value or gross proceeds realized from the ore without deductions of any value the trespasser may have bestowed upon the ore by his labor.

Liberty Bell Gold Mining Co. *v.* Moorhead Mining & Milling Co. (Colorado), 145 Pacific, 686, p. 688, January, 1914.

INNOCENT TRESPASS—MEASURE OF DAMAGES.

In an action for damages for an innocent trespass by mining and milling metalliferous ores, where the trespasser sold the finished product and applied the proceeds to his own use, the owner should recover for his actual loss as his damages are compensatory only; and in such case the measure of damages is the gross value of the ore in place before it was disturbed, and such damages may be ascertained by deducting from the enhanced value or gross proceeds the cost of making the product at the time of conversion.

Liberty Bell Gold Mining Co. *v.* Moorhead Mining & Milling Co. (Colorado), 145 Pacific, 686, p. 688, January, 1915.

PROPERTY HELD IN TRUST.

LEVY AND SALE ON EXECUTION.

Mining property is subject to levy and sale on execution on judgment against a mining company, though the property had been conveyed by a deed of trust by the mining company to trustees named, but the trust created by the deed was not defined, no powers of the trustees were enumerated, and no beneficiary was named, and the conditions imposed upon the mining company had never been performed.

Ohio & Colorado Smelting & Refining Co. *v.* Barr (Colorado), 144 Pacific, 552, p. 553, December, 1914.

MONEY RECEIVED FOR STOLEN ORE BY MINER.

A miner employed by a gold-mining company as a foreman and placed in charge of its sluice boxes, gold dust, and nuggets bears a trust relation to the mining company with respect to the property in his charge and with respect to the gold dust and nuggets in the flumes and sluice boxes within his charge, and the mining company may impress a trust upon the money in the possession of a bank or in the custody of the officer of a court, received by such miner for gold dust and nuggets secreted, appropriated, and stolen by him while acting as such foreman for the mining company and subsequently sold by him and for which he received the money on which the trust was sought to be impressed.

Pioneer Mining Co. *v.* Tyberg, 215 Fed., 501, p. 502.

CONTRACTS OF SALE.

DAMAGES FOR BREACH OF CONTRACT FOR SALE OF COAL.

The breach of a contract by which a coal-mining company agreed to sell and deliver to the purchaser certain stated carloads of coal each day for a certain length of time to be delivered at a stated place gives the purchaser the right to go into the market and purchase other coal to supply the place of that which he failed to receive under the contract, and the measure of damages to which he is entitled in an action for the breach is the difference between the contract price and the market price of coal of similar kind and character at the time and place of delivery, and he would not be entitled to recover anything for the profits he might have made by a resale, because, having purchased coal of a similar kind and character to substitute for that which he failed to receive under the contract, he acquires the profits, upon a resale of the coal bought by him and substituted for the coal which he failed to receive under his contract. But if he made a reasonable and diligent effort to obtain other coal of similar kind and character to supply the place of that he failed to receive under his contract, the damages to which he would then be entitled would be the profits which he could have realized from a resale of the coal. On the other hand, if, on the breach of the contract by the seller, the purchaser made no effort to buy other coal of similar kind and character to supply the place of that he failed to receive under his contract, then the purchaser could recover no damages either for profits or otherwise for the seller's failure to deliver the coal according to the contract.

Log Mountain Coal Co. *v.* White Oak Coal Co. (Kentucky), 174 Southwestern, 721, March, 1915.

CONDITIONAL SALE OF MINING MACHINERY—PRIORITY OF LIENS.

The vendor of mining machinery sold on condition that the title thereto should remain in him until the purchase price was fully paid, has a right and title thereto superior to the lien of a lessor, where the mining machinery was placed in and upon a leased mine for the purpose of its operation, but not intimately embodied in other property of the lessee, though the original contract of sale was not recorded.

Jeffrey Manufacturing Co. *v.* Mound Coal Co., 215 Fed., 222, p. 225.

BINDING EFFECT OF CONTRACT OF PURCHASE—RIGHTS TO SECRET PROFITS.

Persons joining in a contract for the purchase of certain mines and mining properties by which they contracted for designated portions of an equity particularly described in the contract can not be relieved from liability, nor recover a portion of secret profits alleged to have been realized by another joint contractor in connection with the sale of the mines and the mining property to a corporation, where the complaining parties received the precise thing for which they contracted and where the alleged agreement by which the party complained of received the alleged secret profits, neither decreased nor increased the value of the equity described in the original contract, and did not in any manner affect the interest of the complaining parties, and where no fraud or misrepresentation of the value of the property was relied upon.

Ringolsky *v.* Maud L. Mining Co. (Missouri), 171 Southwestern, 56, p. 60, December, 1914.

STATUTORY LIENS.

COAL MINE—NATURE OF DEVELOPMENT WORK.

Labor performed in mining coal in the regular course of operating a mine is not performed in the making of any improvement within the meaning of the statute of Colorado providing for mechanics' liens on mines, and such labor can not be made the basis of a mechanic's or miner's lien.

Expire Coal Co. *v.* Rosa (Colorado), 142 Pacific, 192, p. 193.

VERIFICATION OF STATEMENT FOR LIEN.

The statute of Oregon requires that the statement for a miner's lien shall be verified by the oath of the claimant, or some other person having knowledge of the fact, and it is sufficient for an agent to state in the verification that he has personal knowledge of all the facts set forth in the lien and knows the facts therein set forth to be true, and that the statement is true and correct and that the sums therein named are due the claimant as therein stated.

Loud *v.* Gold Ray Realty Co. (Oregon), 142 Pacific, 785, p. 787.

LESSEE NOT A CONTRACTOR.

A lease of a coal mine in the usual form for the demise of real estate containing provisions for mining and selling coal and paying rent in the nature of royalty is not a contract, nor does it create such a relation between the lessor and lessee that the latter becomes a contractor within the meaning of the mechanics' lien law of Colorado, as under this law a mechanic's lien will not attach to the interests of the owner of a mine for labor done in working or developing the mine where the work is done at the instance of or under contract with one whose only interest is that of a lessee.

Empire Coal Co. *v.* Rosa (Colorado), 142 Pacific, 192, p. 193.

MINER'S RIGHT TO ASSIGN LIEN.

After the lien of a miner has been perfected by the filing and recording of a proper claim of lien, it may be assigned; but under the laws of Oregon, if a miner has a right to perfect a lien and he assigns his debt or claim to another, his assignee takes title to the debt but has no right to perfect the lien.

Loud *v.* Gold Ray Realty Co. (Oregon), 142 Pacific, 785, p. 786.

MINERS WORKING FOR LESSEE—LIENS.

Section 4025 of the Revised Statutes of 1908 (Colorado) makes every contractor, architect, engineer, subcontractor, builder, agent, or other person having charge of the construction, alteration, or repair, either in whole or in part, of any building or other improvement upon or in connection with a mine or its operation, the agent of the owner of the mine for the purposes of mechanics' liens, but this does not apply or include a lessee under an ordinary lease for operating the mine and does not authorize mechanics' liens by miners employed and working for a lessee in the development and operation of a mine under a lease.

Empire Coal Co. *v.* Rosa (Colorado), 142 Pacific, 192, p. 193.

RIGHT OF MINER TO LIEN ON LEASED PROPERTY.

The statute of Colorado provides that mechanics, materialmen, contractors, and all persons of every class performing labor upon or furnishing materials to be used in the construction, alteration, or repair of any building, tunnel, or other improvement upon land shall have a lien upon the property for the value of such services rendered or labor done, whether at the instance of the owner or of any other person acting by his authority or under him as agent, contractor, or otherwise; but this statute does not give a lien to persons employed by a lessee operating a mine under a lease at a stipulated rent in the

nature of royalty, and where the work for which the lien was claimed was done in the ordinary development work under the lease in order to extract the ore.

Grimm *v.* Yates (Colorado), 145 Pacific, 696, p. 697, January, 1915.

NOTICE BY LESSOR NOT REQUIRED TO PROTECT TITLE.

The owner of a mine who leases it to be operated under the terms of the lease by the lessee, where none of the work performed by the lessee was authorized by the lease itself but only contemplated as a part of the operation of the mine, is not required to give the notice provided for in the statute of Colorado in order to protect his title and the property in the mine from mechanics' and miners' liens authorized by the statute.

Grimm *v.* Yates (Colorado), 145 Pacific, 696, p. 698, January, 1915.

MINERS EMPLOYED BY LESSEE—FAILURE OF OWNER TO GIVE NOTICE.

Miners performing services and persons furnishing supplies for the mine at the request of the lessee have a right to perfect and hold liens upon a mine for such labor or supplies unless the owner of the mine has posted at not less than three conspicuous places upon the mine, at or near the place where the same was worked, a signed written notice, stating the name of the lessee or contractor, and that the owner will not be responsible for any debts contracted by the lessee or contractor in connection with the operation of the mine.

Loud *v.* Gold Ray Realty Co. (Oregon), 142 Pacific, 785, p. 787.

ENFORCING LIENS—FAILURE TO GIVE NOTICE—PLEADING.

Section 1149, Remington and Ballinger Code of the State of Washington, gives every person performing labor for a person or corporation in the operation of any mining or manufacturing company a lien on the real and personal property of such person or corporation in the operation of its business, and in an action to foreclose a lien on mining machinery the mine operator can not after judgment and on appeal raise the objection or take advantage of the failure of the lienor to serve the statutory copy of notice of lien upon him, as such a question must be raised in the trial court, and defects in pleadings that might have been cured by amendment will not be considered after verdict.

Cook *v.* Snyder (Washington), 146 Pacific, 156, p. 157, February, 1915.

OPTION AGREEMENT TO PURCHASE MINE—RIGHT OF MINERS TO LIEN.

Section 2904 of the Revised Statutes of Arizona, 1901, provides that all miners who may labor upon any mine or mining claim shall have a lien upon the same for any amount due them; and the amend-

ment to this section, as incorporated in section 3654 of the Civil Code of 1913, provides that miners who labor in or upon any mine or mining claim shall have a lien for the sum due and that such lien shall attach to the mine or mining claim under either of the following conditions: (1) By virtue of a contract between the miner and the owner of the mining claim or his agent; (2) by virtue of a contract between the miner and the lessee of any such mine or his agent where by the terms of the lease the lessor permits the lessee to develop or work the mine; and (3) where by virtue of a contract between the miner and any person or corporation having an option to buy or contract to purchase the mine, where such option contract permits the purchaser to work or develop the same; but these statutes do not authorize a miner performing labor upon a mine for a purchaser under an option agreement to purchase the same, to enforce his lien against the mining property, though such option contract to purchase the mine permitted him to enter on and develop the property, where such option contract was given before the amended act was passed, as such option contract does not make the purchaser an agent of the owner.

Foltz *v.* Noon (Arizona), 146 Pacific, 510, February, 1915.
Oceanic Gold Mining Co. *v.* Steinfeld (Arizona), 147 Pacific, 717, April, 1915.

ENFORCING LIEN AGAINST PROPERTY OF LESSORS.

Miners employed by the lessee of a mine who operates the same and who holds the optional agreement to purchase the machinery and equipment which, with the mine, was leased for its operation, may enforce their lien against the leasehold interest of the lessee and against the machinery so held by the lessee under his option to purchase, though he never exercised such option and never became the absolute owner of the machinery and personal property, where the lessor failed to have such option agreement recorded within 10 days after the change of the possession of the property as required by the statute of Washington, as in such case the sale must be treated as absolute as to the liens of the miners, who were subsequent creditors in good faith.

Cook *v.* Snyder (Washington), 146 Pacific, 156, p. 157, February, 1915.

RETROACTIVE EFFECT OF STATUTE—LESSEE NOT AGENT OF LESSOR.

The right of a miner to a lien upon a mine or mining claim for labor done or material furnished under the statute of Arizona must be tested by the statute existing at the time the labor was performed or the materials furnished; and if the labor was performed or the materials furnished before the amended act of December, 1912, then the right to the lien must be found as the statute existed before that

date, as the amendatory act can not have a retroactive operation, as such an application of the amendatory act would be a substantial interference with the obligation of the contract itself; and prior to the amendatory act the mere fact of the existence of the contract to purchase or an option to buy a mining claim, by the terms of which the purchaser was permitted to enter upon, work and develop the mine, with no additional powers over the property, would not constitute the proposed purchaser the agent of the owner, and would not make the mine or mining property subject to the lien of miners who were employed by the person holding the contract or option to purchase.

Foltz *v.* Noon (Arizona), 146 Pacific, 510, February, 1915.
Oceanic Gold Mining Co. *v.* Steinfeld (Arizona), 147 Pacific, 717, p. 718, April, 1915.

DAMAGES FOR INJURIES TO MINERS.

ELEMENTS OF DAMAGES.

DOUBLE DAMAGES NOT AUTHORIZED.

In an action by a miner for damages for injuries caused by the negligence of a mine operator, a miner is not entitled to recover for the loss of time occasioned by the injury and for impairment of his power to earn money during the same period of time, as this would be authorizing a recovery of double damages.

Main Jellico Mountain Coal Co. *v.* Young (Kentucky), 169 Southwestern, 841, p. 842, October, 1914.

ALLEGATIONS AS TO LOSS OF TIME.

In an action by a miner for injuries caused by a fall of slate from the roof where the petition alleged that the slate cut off the plaintiff's thumb, injured his shoulder, caused him great pain and physical suffering, and partially destroyed his power to earn money, and that the injuries are permanent in character, a recovery for time lost by the injured miner is not justified, in the absence of any claim or plea therefor in his petition.

Main Jellico Mountain Coal Co. *v.* Young (Kentucky), 169 Southwestern, 841, p. 842, October, 1914.

WRONGFUL DEATH OF CHILD—AGGRAVATING CIRCUMSTANCES.

In an action by a parent for the wrongful death of a minor son caused by the alleged negligence of a smelting and refining company an instruction is erroneous which ignores and makes no mention of the circumstances, whether aggravating or mitigating, under which the killing took place, and leaves to the jury as a simple question of dollars and cents to say by what amount the parent has suffered actual pecuniary loss in the death of a child, as such a construction makes the provision of the statute wholly meaningless and inoperative

which provides that damages shall be assessed "having regard to the mitigating or aggravating circumstances attending such wrongful act, neglect, or default;" and under this provision the jury are not restricted to such damages as are the necessary and inevitable result of the injuries sustained, and are free also to consider, in making up their verdict, the circumstances attendant on the wrongful act, neglect, or default, which gave rise to the injury complained of.

Dalton v. St. Louis Smelting & Refining Co. (Missouri Appeals), 174 Southwestern, 468. p. 471, March, 1915.

DEATH OF HUSBAND—MEASURE OF RECOVERY.

In an action by a widow for damages for the wrongful death of her deceased husband, it is proper for a court to instruct the jury to the effect that the measure of damages is the detriment which the widow proximately sustained by reason of the death of the husband, and in assessing the damages it is the duty of the jury to consider the age of the deceased, the number of years he might have reasonably and probably been expected to perform manual labor, his contribution to and support of the plaintiff and his infant children, and all the circumstances and conditions surrounding their relations, and the likelihood and probability of the deceased to contribute to the maintenance and support of the plaintiff, and to fix compensation for the injuries shown to have been inflicted on the plaintiff by reason of the death of the deceased; and such a construction is not open to the charge of error in failing to limit the plaintiff's recovery and pecuniary damages.

Great Western Coal & Coke Co. v. Coffman (Oklahoma), 143 Pacific, 30, p. 33.
Great Western Coal & Coke Co. v. McMahan (Oklahoma), 143 Pacific, 23.
Great Western Coal & Coke Co. v. Cunningham (Oklahoma), 143 Pacific, 26.

DAMAGES NOT EXCESSIVE.

INSTANCES.

A judgment for $3,000 for injuries that are not only serious in their nature, but of a permanent character, and will follow the miner through life, seriously impairing his capacity to perform the physical labor upon which he must depend for his support, and where at the time of receiving the injuries he was but 29 years of age, is not excessive.

Darby Coal Mining Co. v. Shoop (Virginia), 83 Southeastern, 412, p. 416, November, 1914.

A judgment for $7,500 for the death of a miner who was 28 years of age, in good health and performing manual labor regularly, with an earning capacity of $125 per month, leaving a wife and two children; and where it appeared that during the four years of his married life he had accumulated about $500.

Great Western Coal & Coke Co. v. Coffman (Oklahoma), 143 Pacific, 30, p. 34.

A verdict for $15,000 for the death of a miner 43 years old leaving a widow and six children and earning as high as $10 a day, and where he was shown to be sound of body and mind, was able to work all the time, and all his earnings went to the support of his family.

San Bois Coal Co. *v.* Resetz (Oklahoma), 143 Pacific, 46, p. 50.

A verdict of $7,334 for an injury to a coal miner can not be said as a matter of law to be excessive where it appears that a piece of slate some 5 feet in length, 2½ feet in width, and 6 to 8 inches thick in the center, fell upon the miner's back and crushed him to the ground, and where the evidence showed that the spinous processes of the backbone were broken down; that the muscles of the back on either side of the backbone had become rigid and hardened; that the lower limbs of the miner had become benumbed; and where the weight of the miner had been reduced from 150 pounds to 124 pounds, and he suffered from the disarrangement of the proper action of his kidneys and bowels, and where it appeared that at the time of the injury he was a stout man, able to earn $4 or $5 a day, and since the injury has been entirely incapacitated from doing any kind of physical labor.

New Bell Jellico Coal Co. *v.* Sowders (Kentucky), 172 Southwestern, 914, p. 917, January, 1915.

A verdict and judgment for $9,000 in favor of an able-bodied boy 18 years of age, can not be held as a matter of law to be excessive where the boy's foot was mashed by being run over by a car, the bone of the leg broken, the flesh lacerated and torn from it, and where 27 pieces of the bone protruded and were removed, and where the surgeon described the injury as a compound fracture with extensive lacerations of the tissues upon the ankle and knee, and that the bone was broken and exposed and the ankle joint stiff and one leg rendered shorter than the other, and where, before the wound healed, parts of the bone disintegrated and the patient suffered excruciating pain.

Dillon *v.* United States Coal & Coke Co. (West Virginia), 84 Southeastern, 956, p. 960, April, 1915.

QUARRY OPERATIONS.

STORING EXPLOSIVES—NUISANCE.

A person is guilty under the law of maintaining a nuisance who keeps stored indefinitely in a thickly settled neighborhood large amounts of high explosives which are liable, as dynamite is liable, to explode and do serious injury to surrounding persons and property, though the manner of keeping such explosives is characterized by no special negligence.

Sloss-Sheffield Steel & Iron Co. *v.* Prosch (Alabama), 67 Southern, 516, p. 519, December, 1914.

STORING EXPLOSIVES IN DANGEROUS PLACE.

A corporation engaged in operating iron-smelting furnaces and in quarrying rock, is liable for damages caused by an explosion and is guilty of maintaining a nuisance where it kept explosives stored in a wooden building or magazine situated near and in dangerous proximity to a thickly settled community and close to a railroad track owned and operated by the corporation and close to certain large slag piles, where hot slag was deposited by the corporation in the course of the operation of its furnaces and where hot slag was carried from its furnaces close by such explosives by engines hauling hot pots containing slag, as the place of storing the explosives was one where the explosion might reasonably be expected to occur; and such act of keeping and storing such explosives is sufficient to create a prima facie liability for injuries resulting from an explosion of such explosives.

Sloss-Sheffield Steel & Iron Co. *v.* Prosch (Alabama), 67 Southern, 516, p. 518, December, 1914.

USE OF EXPLOSIVES—DUTY AS TO USE.

The value of dynamite rests exclusively in its quality as a powerful explosive and if a quarry owner is under the necessity of using dynamite, or other dangerous explosives, the law casts the duty upon him to keep, handle, and use them in a reasonably safe and careful manner.

Sloss-Sheffield Steel & Iron Co. *v.* Prosch (Alabama), 67 Southern, 516, p. 518, December, 1914.

WORKMAN NOT REQUIRED TO INSPECT—RISKS NOT ASSUMED.

A workman engaged in a stone quarry, working under or near a bank of a gravelly nature upon which there was more or less stone, and which arose perpendicularly for about 10 feet and then continued for some 35 feet at an angle of about 45° is not required to examine the top of the bank above the place where he was at work; and while he was bound to exercise ordinary care to prevent injury, yet the responsibility for the condition of the sides and top of the bank, that which the workman could not see, was upon the quarry operator. A workman put to work in a place which subjects him to the danger of being injured by sliding rock or earth, does not assume the risk where the danger is not obvious to him, and where the master, with actual or constructive knowledge or notice of the danger, fails to warn him or to exercise due care in making the place safe.

Novy *v.* Breakwater Co. (Connecticut), 92 Atlantic, 668, p. 671, December, 1914.

OPERATOR'S KNOWLEDGE OF DANGER—DUTY TO INSPECT.

The owner and operator of a stone quarry using dynamite in loosening the stone under which employees were working, is charged with knowledge that parts of the ledge may at any time drop down on the workmen, and accordingly he should have the loose stone removed; and employees may assume that such duty had been performed by the employer. Under such circumstances the duty of inspection devolved upon the employer, and where the proper inspection would have disclosed the loose rock, the employer is liable for an injury to an employee caused by a fall of such loose rock.

Paul Stone Co. *v.* Saucedo (Texas Civil Appeals), 171 Southwestern, 1038, p. 1039, January, 1915.

PROVIDING SAFE PLACE—EMPLOYEE MAY ASSUME DUTY PERFORMED.

Under the rule that the operator of a quarry must exercise reasonable care to furnish the workmen in the quarry a reasonably safe place in which to work, the employees may, in the absence of knowledge or information to the contrary, assume that the operator had performed his duty in this respect.

Losasso *v.* Jones Brothers' Co. (Vermont), 93 Atlantic, 266, p. 270, February, 1915.

NEGLIGENCE OF VICE PRINCIPAL—LIABILITY OF OPERATOR.

The foreman in a quarry who has full charge of its operations and intrusted with the performance of the duties resting upon the owner or operator is a vice principal, and notice to him by a powder man immediately after a blast of the dangerous condition of a header is in law notice to the owner or operator, and the owner or operator is liable for the foreman's negligence in not remedying the defect so as to make reasonably safe the working place where employees are to proceed to work.

Losasso *v.* Jones Brothers' Co. (Vermont), 93 Atlantic, 266, p. 270, February, 1915.

DUTY OF EMPLOYER TO FURNISH MEDICAL AID TO INJURED EMPLOYEES.

The rule that whenever one person employs another to perform dangerous work, and while performing such work the employee is so badly injured as to incapacitate him for caring for himself, the duty of providing medical treatment is devolved upon the employer, based on the unexpressed humane and natural understanding existing between parties to the effect that whenever an employee is so injured that he can not care for himself, then the employer will furnish him medical or surgical treatment, must be applied in the case of the operation of a quarry where an employee in the quarry was so grievously hurt that he was in great danger of bleeding to death and could not assist himself or procure the necessary medical aid.

Hunicke *v.* Meramec Quarry Co. (Missouri), 172 Southwestern, 43, p. 54, December, 1914.

PUBLICATIONS ON METHODS OF MINING.

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of especial interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 45. Sand available for filling mine workings in the northern anthracite coal basin of Pennsylavnia, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls., 7 figs.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs.

BULLETIN 60. Hydraulic mine filling; its use in the Pennsylvania anthracite fields; a preliminary report, by Charles Enzian. 1913. 77 pp., 3 pls., 12 figs.

BULLETIN 69. Coal mine accidents in the United States and foreign countries, compiled by F. W. Horton. 1913. 102 pp., 3 pls., 40 figs.

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TECHNICAL PAPER 21. The prevention of mine explosions, report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 30. Mine-accident prevention at Lake Superior iron mines, by D. E. Woodbridge. 1913. 38 pp., 9 figs.

TECHNICAL PAPER 41. Mining and treatment of lead and zinc ores in the Joplin district, Mo., a preliminary report, by C. A. Wright. 1913. 63 pp., 5 figs.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912; compiled by F. W. Horton. 1913. 74 pp., 10 figs.

TECHNICAL PAPER 58. The action of acid mine water on the insulation of electrical conductors; a preliminary report, by H. H. Clark and L. C. Ilsley. 1913. 26 pp., 1 fig.

TECHNICAL PAPER 59. Fires in Lake Superior iron mines, by Edwin Higgins. 1913. 34 pp., 2 pls.

TECHNICAL PAPER 61. Metal-mine accidents in the United States during the calendar year 1912, compiled by A. H. Fay. 1913. 76 pp., 1 fig.

TECHNICAL PAPER 92. Quarry accidents in the United States during the fiscal year 1913, compiled by A. H. Fay. 1914. 45 pp.

Bulletin 102

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

THE INFLAMMABILITY OF ILLINOIS COAL DUSTS

BY

J. K. CLEMENT AND L. A. SCHOLL, JR.

ILLINOIS COAL-MINING INVESTIGATIONS
COOPERATIVE AGREEMENT

[This report was prepared under a cooperative agreement with the Illinois State Geological Survey and the department of mining engineering of the University of Illinois.]

WASHINGTON
GOVERNMENT PRINTING OFFICE
1916

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First edition. December, 1915.

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THE INFLAMMABILITY OF ILLINOIS COAL DUSTS.

By J. K. Clement and L. A. Scholl, Jr.

INTRODUCTION.

Among the problems investigated by the Bureau of Mines the coal-dust problem has received much attention. Several of the bureau's publications deal specifically with the subject. Bulletin 20[a] contains an account of preliminary experiments of the bureau to determine the explosibility of coal dusts as well as a review of pertinent literature. Bulletin 50[b] is a report of a laboratory study of the inflammability of coal dust. The first series of coal-dust explosion tests in the bureau's experimental mine at Bruceton, Pa., is described in Bulletin 56[c], and Technical Paper 56[d] presents notes on the prevention of dust and gas explosions in coal mines, and Technical Paper 84[e] discusses the causes of such explosions and describes methods and apparatus devised by engineers of the bureau for preventing and checking them.

The present paper is a report of a detailed study of the bituminous dusts of Illinois mines, and is a part of the investigations conducted by the bureau in cooperation with the Illinois State Geological Survey and the department of mining engineering of the University of Illinois.[f]

It is well known that bituminous coals like those found in Illinois yield dusts that may give rise to explosions of great violence. In order that proper methods might be devised for preventing dust explosions in Illinois mines it was considered desirable to obtain accurate information on the quantity and character of these dusts. Experiments and tests of dust explosions in an experimental mine as

[a] Rice, G. S., The explosibility of coal dust with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp.

[b] Frazer, J. C. W., Hoffman, E. J., and Scholl, L. A., jr., A laboratory study of the inflammability of coal dust. 1913. 60 pp.

[c] Rice, G. S., Jones, L. M., Clement, J. K., and Egy, W. L., First series of coal-dust explosion tests in the experimental mine. 1913. 115 pp.

[d] Rice, G. S., Notes on the prevention of gas and dust explosions in coal mines. 1913. 24 pp.

[e] Rice, G. S., and Jones, L. M., Methods of preventing and limiting explosions in coal mines. 1915. 50 pp.

[f] A statement of the plan of organization and of the scope of the work being carried on under the cooperative agreement is contained in the "Preliminary Report on Organization and Method of Investigations, Illinois Coal Mining Investigations Cooperative Agreement," University of Illinois, Urbana, 1913.

well as in a large steel gallery comparable in area of cross section to mine entries have been made by the Bureau of Mines. Similar experiments have been made in Austria, England, France, and Germany. Experiments under conditions similar to those of actual mining practice are obviously more conclusive than tests made in a laboratory. On account, however, of the expense, time, and quantity of dust required, it is not practicable to conduct large-scale tests in studying a large number of dusts and recourse must be had to laboratory apparatus and methods.

Laboratory devices for the study of coal-dust explosions have been devised by a number of investigators. Most of these were intended for qualitative tests only. Quantitative methods have been devised by investigators of the English and French mining-experiment stations.[a] In the experiments described in this report, laboratory tests of inflammability were made with an apparatus devised by Dr. J. C. W. Frazer,[b] formerly a chemist of the Bureau of Mines.

Laboratory tests of inflammability have been made on more than 500 samples of dust from 100 representative Illinois mines. As a result of these tests it may be stated that:

1. The coals of Illinois when ground fine enough yield highly inflammable dusts which when suspended in air may give rise to violent explosions.

2. In the majority of the mines of the State the dusts adhering to the ribs are inflammable. In about one-fourth of the mines the quantity of rib dusts present in the entries is sufficient to form an explosive mixture with air.

3. Most of the road dusts are inflammable when ground to a sufficient degree of fineness.

METHODS OF COLLECTING DUST SAMPLES.

The dust samples were collected by mining engineers of the University of Illinois and of the Bureau of Mines. Preliminary preparation of the samples and moisture determinations were made at Urbana, Ill., under the direction of Prof. S. W. Parr, by J. W. Lindgren and F. C. Whithen. The dust laboratory was equipped by the mining department of the University of Illinois, and the inflammability tests were made by L. A. Scholl, jr., junior chemist of the Bureau of Mines, and G. W. Porter of the mining department of the university, under the direction of Messrs. Frazer and Clement of the Bureau of Mines.

[a] Taffanel, J., and Durr, A., Cinquième série d'essais sur les inflammations de poussières; Essais d'inflammabilité, 1911, 70 pp.; Taffanel, J., Note sur des expériences relatives au classement des gisements de poussières, 1912, 15 pp; Second Report to the Secretary of State for the Home Department of the Explosions in Mines Committee, London, 1912, 43 pp.

[b] See Frazer, J. C. W., Hoffman, E. J., and Scholl, L. A., jr., A laboratory study of the inflammability of coal dust: Bull. 50, Bureau of Mines, 1913, pp. 5–7.

FIGURE 1.—Map of Illinois showing districts into which State was divided for investigation.

For the purposes of the cooperative mining investigation the State was divided into eight districts, the mines working the same seam being grouped as far as possible. The boundaries of the districts are shown in figure 1. In the eight districts 100 representative shipping mines were chosen for examination. Figure 2 shows the principal workable coal seams in the State. Inasmuch as the 100 representative mines selected for examination represent 25 per cent of the mines of the State that ship coal, it is thought that any generalizations and conclusions based on results obtained in a study of those mines should be representative of all the mines of the State.

COLLECTION OF FACE SAMPLES.

The method of sampling was essentially the same as the method described in Bureau of Mines Technical Paper 1.[a] One difference was that the sample cut from the face was ground in a special grinder to ⅛-inch size or smaller. After the sample had been ground, it was reduced by means of a mechanical riffle to a sample weighing 3 to 5 pounds, which was placed in an air-tight can with a screw cap. The cap was wound with tape as an additional safeguard against deterioration of the sample. It was thought that the use of the special grinder and the riffling device would eliminate in part at least the personal equation. Three face samples were taken in the same manner from three places in each mine, and at some of the more important mines, and at least at one mine in each district, six face samples were taken at different points in order to ascertain the variations or uniformity in the chemical composition of the coal in the particular mine and bed of coal.

The inflammability tests were conducted as soon as possible after the receipt of the sample at Urbana. This precaution was deemed advisable because previous investigations had shown that Illinois coals lose inflammable gases during the first two weeks after mining.[b]

Before an inflammability test was made the sample was subjected to the following preliminary treatment: The sample after removal from the container was placed in a shallow tray, weighed, placed in a drying oven, and dried at a temperature of about 35° C. to constant weight, a current of air being forced over the tray of coal by a small fan. The sample was then ground fine in a grinder similar to the one employed in the mine, reduced by riffling to a sample of about 500 grams, and passed through a 60-mesh sieve. Two small 60-gram samples of this 500-gram sample of 60-mesh coal were placed in separate bottles, one sample for analysis and one for inflammability tests. The rest of the 500-gram sample was placed in an air-tight glass fruit jar as a reserve sample. From the three individual face

[a] Holmes, J. A., The sampling of coal in the mine: Tech. Paper 1, Bureau of Mines, 1911, 18 pp.

[b] Parr, S. W., and Barker, Perry, The occluded gases in coal: University of Illinois, Bull. 32, 1909, p. 27.

samples to be used in the inflammability tests a composite sample was made, 15 grams being taken from each 60-gram sample by quartering after a thorough mixing. The composite sample was then ground to pass through a 200-mesh sieve, placed in a bottle, and labeled. As soon as possible after grinding, the inflammability tests of the finely ground sample were made. In addition to the composite samples from each mine, general composite samples for each district were made by taking a definite quantity, usually 5 grams, from the finely ground individual composite samples.

In this report for convenience the coal beds are designated as "No. 1," "No. 2," etc., after the practice of the Illinois State Geological Survey. The equivalent designations adopted by the United States Geological Survey are given below:

Number as used by State geological survey.	Designation adopted by United States Geological Survey.
No. 1	No. 1; term is applied to several discontinuous beds.
No. 2	Murphysboro.
No. 3	Harrisburg, except in Springfield and Peoria districts, where designation is Springfield.
No. 6	Herrin (displaces term "Belleville").
No. 7	No. 7.

COLLECTION OF MINE-DUST SAMPLES.

The mine dusts collected comprised two kinds—rib dusts and road dusts. The rib-dust samples were gathered from the ribs of the entries and rooms, and the road-dust samples were collected from the floors of the entries.

The rib-dust samples were taken at three different points in each mine, as follows: Sample 3 on the main haulage entry, sample 2 on the secondary haulage entry, and sample 1 inside the last crosscut near the face of the entry or from a room. The method employed in obtaining the samples was as follows: A chalk line was drawn from the roof to the floor and the dust was brushed from the rib onto a sheet of clean paper with a 1-inch varnish brush until sufficient dust was obtained to fill a 40-gram glass bottle. The sample was immediately transferred to the bottle, which was closed with a rubber stopper. The height and the width of the rib brushed were measured in order to have an estimate of the amount of rib dust present in the entries throughout the mine, and the relative humidity and the volume of air from the intake were observed.

Samples of road dust were taken at two different points in each mine—one (sample 2) on the main haulage entries near the point at which the corresponding rib-dust sample was taken; the other

(sample 1) on the secondary haulage entries at a place near the point at which the corresponding rib-dust sample was obtained. A section of entry about 100 feet long was selected for sampling, the place where the rib-dust sample was taken being about the middle of this distance. With a small metal spatula, a series of samples approximating about 1 ounce for each 3 feet of entry were taken. The dust collected in this manner was placed in a metal can provided with a screw cap and having a minimum capacity of 3 pounds. The cap was wound with tape as an extra precaution, and the can was forwarded to the laboratory.

PRELIMINARY TREATMENT OF MINE-DUST SAMPLES.

As the treatment of the road-dust and of the rib-dust samples in the laboratory preliminary to the inflammability determinations was the same, it is discussed under one heading. Upon the receipt of cans of road dust and bottles of rib dust at the laboratory, they were opened and the dust air dried on a tray in the drying oven at a temperature of 35° C. until a constant weight was obtained and the loss in weight determined. Each dust sample was separated into parts by being screened with 20, 60, 100, and 200 mesh sieves and the several parts were weighed.

In inflammability tests of rib and road dusts three classes of samples were used, as follows: (1) The part of the air-dried sample passing through a 20-mesh sieve; (2) the part passing through a 200-mesh sieve; (3) samples prepared from the part of the air-dried sample that passed the 20-mesh sieve after grinding to pass a 200-mesh sieve.

It might well be expected that the inflammability of an air-dried sample of dust would be different from that of a sample containing its original moisture. Experiments have shown that although as a rule the air-dried sample develops slightly higher pressure than the "as-received" sample the difference is negligible.

METHOD OF ANALYZING DUST SAMPLES.

Upon the completion of the inflammability tests of a sample of dust it was immediately shipped to the Bureau of Mines testing station at Pittsburgh, where analyses were made under the direction of A. C. Fieldner, chemist. Ultimate analyses and calorimeter tests were made of the face samples, but proximate analyses and calorimeter tests only were made of the road and rib dusts.

PROXIMATE ANALYSIS.

Moisture.—A 1-gram sample of fine coal in a porcelain capsule is heated for 1 hour at 105° C. in a constant-temperature oven through which dry air is circulated. The figure representing loss in weight

multiplied by 100 represents the percentage of moisture in the sample of air-dried coal.

Volatile matter.—A 1-gram sample of fine coal is heated to a temperature of 950° C. in a 10 c. c. platinum crucible with a platinum capsule cover in an especially designed electric furnace for 7 minutes.[a] The loss in weight from heating less the weight of the moisture determined at 105° C. multiplied by 100 gives the percentage of volatile matter in the sample.

Ash.—The porcelain capsule containing the residue from the moisture determinations is placed in a muffle furnace and heated slowly, the volatile matter being driven off; finally the temperature is raised to 750° C., and heating is continued until the weight becomes constant. The weight of the residue in the crucible is the ash or ignited mineral matter in the coal, which, multiplied by 100, gives the percentage of ash in the coal.

Sulphur.—The sulphur is determined by the Eschka method of heating 1 gram of the fine coal with 2 grams of "Eschka mixture" (MgO 2 parts and Na_2CO_3 1 part) in a 40 c. c. porcelain crucible. The crucible is heated slowly at first to avoid losses, the heat being raised until all black particles have been consumed. After digestion with hot water, filtration, acidification with HCl, and addition of bromine water, the sulphur, which is in the form of the sulphate, is precipitated as barium sulphate, filtered, ignited, and weighed. The weight of barium sulphate multiplied by 13.74 gives the percentage of sulphur in the sample.

Fixed carbon.—The difference between the sum of the percentages of the moisture, ash, and volatile matter in a sample of fine coal and 100 is called the percentage of fixed carbon in the sample.

ULTIMATE ANALYSIS.

Carbon and hydrogen.—The percentages of carbon and hydrogen in a sample of fine coal are determined by the combustion method, whereby the carbon and the hydrogen are converted to carbon dioxide and water, respectively, and, after absorption by a suitable medium each is determined by the increase in weight of the absorbing medium. A sample weighing 0.2 gram is employed. Calcium chloride is used to absorb the water formed by the combustion, and the increase in weight of the calcium chloride tube multiplied by 55.95 equals the percentage of hydrogen in the sample. Potassium hydroxide is used to absorb the carbon dioxide found as a result of the combustion, and the increase in weight of the potassium hydroxide bulbs multiplied by 136.36 gives the percentage of carbon in the sample.

[a] See Fieldner, A. C., Notes on the sampling and analysis of coal: Tech. Paper 76, Bureau of Mines, 1914, p. 20.

Nitrogen.—For determining the nitrogen content of the sample of fine coal the Kjeldahl-Gunning method is employed. The nitrogen in a 1-gram sample is converted to ammonium sulphate by boiling a mixture of the coal, concentrated sulphuric acid, potassium sulphate, and a drop of mercury in a round-bottomed flask. After treatment to precipitate the mercury and take care of the excess acid, the solution is made alkaline with NaOH and the ammonia is distilled into a measured amount of standard sulphuric-acid solution to which cochineal indicator has been added. The nitrogen content of the sample is then calculated after the excess sulphuric acid has been titrated with a standard solution of NH_4OH.

Calorific value.—The calorific value of the fine coal is determined by means of the Mahler bomb calorimeter. One gram of coal is burned in the bomb. From the rise in temperature of the calorimeter system the heating value of the coal dust is calculated.

For a complete description of the methods and the apparatus employed in analyzing the dusts the reader is referred to Technical Paper 8 of the Bureau of Mines[a].

CHARACTER AND OPERATION OF APPARATUS USED IN TESTS.

The equipment of the laboratory is shown in Plate I, *A*. The electric current for the heating coil was furnished by a motor generator giving direct current at 110 volts and provided with a voltage regulator.

DESCRIPTION OF APPARATUS.

The inflammability apparatus used in the investigations is shown in Plate I, *B*, and figure 3.

The apparatus consists of an explosion flask, a device for putting the dust in suspension, a platinum ignition coil, and a device for measuring the pressure developed in the explosion flask.

The explosion flask *a* (fig. 3) is made of heavy glass with a low coefficient of expansion, and has a capacity of 1,500 c. c. At the top and the bottom are large tubulures which are ground true. A wide rubber band placed around the lower tubulure, and extending a short distance below it, contracts over the end of the tubulure so as to form a cushion between the tubulure and the brass plate *c*. A rubber gasket is cemented to the under side of the brass plate *k*, where it rests on the upper tubulure. By this means, when the nuts above the steel piece *l* are screwed down, the joints at the top and the bottom of the flask are made gas tight.

The dust injector consists of a small glass funnel, *b*, cemented into the brass plate *c*. A piece of 30-mesh copper gauze covers the top

[a] Stanton, F. H., and Fieldner, A. C., Methods of analyzing coal and coke: Tech. Paper 8, Bureau of Mines, 1913, 42 pp.

of the funnel and serves to break up the dust into a cloud when the dust is ejected from the funnel into the flask by the release of the compressed air contained in the 150 c. c. glass bulb *d*. The igniter *i* consists of a coil of 100 cm. of No. 26 platinum wire wound upon quartz insulators which are attached to the heavy nickel leads, *j*. The nickel leads pass the wooden bushings in the top brass plate and hold the coil in the center of the explosion flask. The ends of the platinum wire are soldered with silver to the nickel leads.

The device used to measure the pressure developed in the flask comprises a small 50 c. c. flask, *q*, containing a weighed amount of mercury, and a small steel ball, *n*, which is ground to fit practically

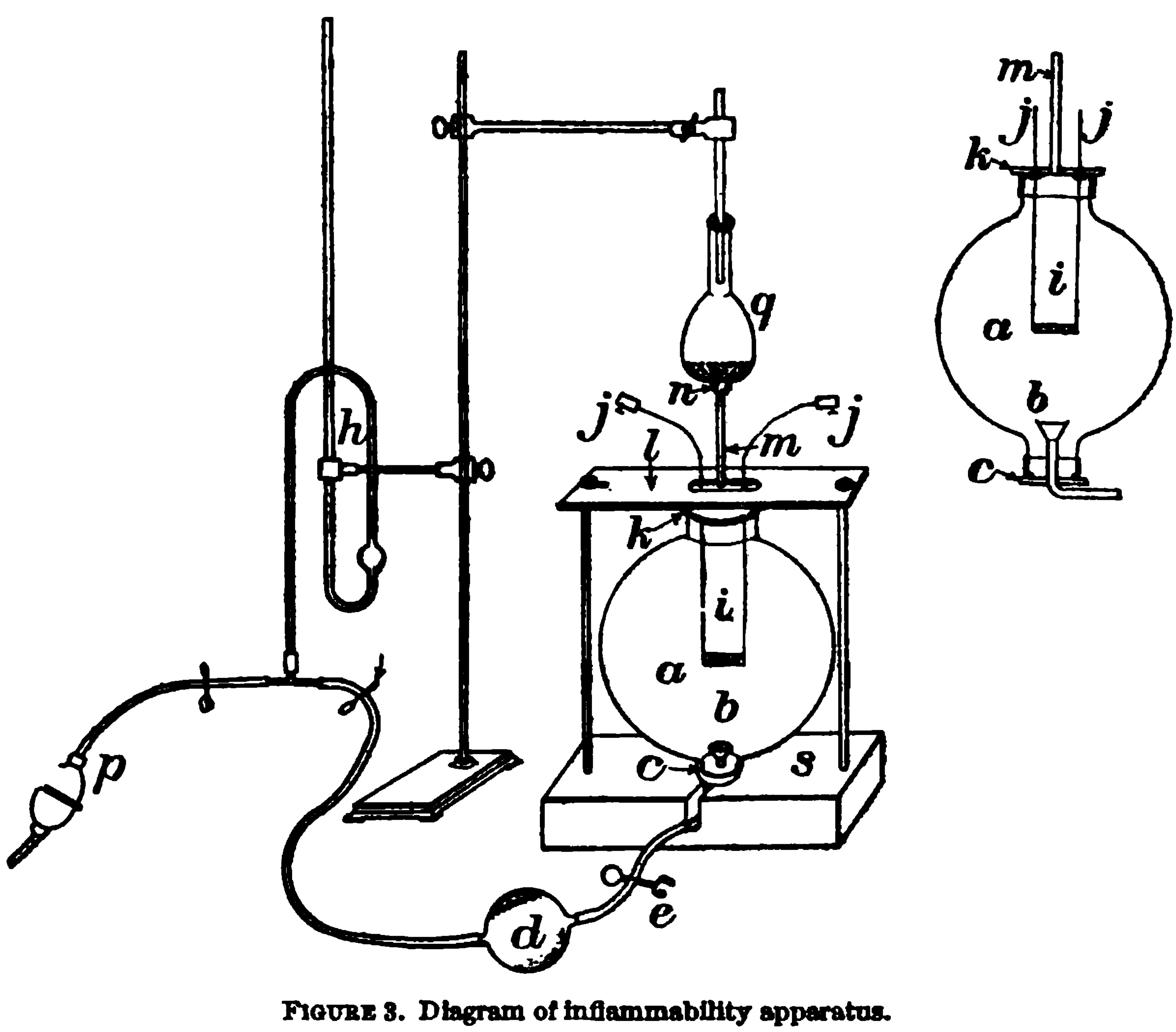

FIGURE 3. Diagram of inflammability apparatus.

gas-tight into the brass tube *m*, which communicates through *k* with the large flask *a*.

OPERATION OF APPARATUS.

A weighed amount of dust (0.05 gram) is placed in the funnel, which is then connected to the bulb *d* by means of a short piece of rubber tubing closed with a pinch cock, *e*, and placed in a shaped receptacle in the wooden block *s*. By means of the compression bulb *p* the air in *d* is compressed to a pressure of 200 mm. of mercury as indicated by the manometer *h*.

With the apparatus assembled as shown in Plate I, *B*, the desired current is passed through the platinum coil for exactly 3 minutes, and

A. EQUIPMENT OF LABORATORY AT URBANA, ILL.

B. INFLAMMABILITY APPARATUS AND ACCESSORIES.

during that interval the expanding air in the flask *a* (fig. 3) is released by the lifting of the steel ball *n* at intervals of 1, 2, and 2¾ minutes. At the expiration of the third minute the pinch cock is opened and the dust in the funnel is blown into the flask. To prevent release of pressure through *b* it has been found desirable to introduce a check valve.

The experiment is repeated, the weight of mercury being varied each time until the pressure lies between two weights 5 grams apart. The mean of the two values is then accepted as the maximum pressure that each dust being studied is able to produce at the temperature used. The maximum pressure that each dust will develop is determined in the above manner for five different temperatures—800°, 900°, 1,000°, 1,100°, and 1,200° C. In all the experiments 0.05 gram of dust is used, and the resultant pressure checks consistently if identical conditions are maintained for all experiments. It is essential to have a steady current, and for this purpose an ammeter is always in circuit, the current being regulated closely by means of a rheostat.

CALIBRATION OF PLATINUM COIL.

As the character of the explosion and the pressure developed by it depend on the temperature of the platinum coil or grid, the determination of this temperature is important. The temperature is calculated from the electrical resistance of the wire as measured by the fall-of-potential method. To determine the temperature-resistance curve of the platinum wire, the wire is wound on a porcelain tube and heated in an electric furnace, the resistance being measured by a Wheatstone bridge. The temperature is measured by a thermocouple whose hot junction is in the axis of the porcelain tube midway between the ends of the platinum spiral. Proper corrections are made for the resistance of the leads.

On account of the cooling effect of the leads the temperature of the platinum coil is greater in the middle than at the ends, and the temperature determined by the resistance of the wire is somewhat below the actual temperature of the middle part of the coil.

The coating of ash deposited on the wire by the coal dust produces a change in the electrical resistance of the wire as well as in the radiating surface. From time to time the wire is cleaned by removing it from the grid and immersing it in hydrofluoric acid. The change in the resistance of the wire occasioned by impurities as well as by variations in the dimensions of the wire was so rapid that it is found necessary to verify the temperature-resistance curve once a day. Verification is accomplished by determining the resistance of the coil at the so-called "critical temperature" for Pittsburgh standard dust. The resistance of the coil at the critical temperature and the temperature coefficient of the wire being known, the corrections to the temperature-resistance curve are readily calculated.

INFLAMMABILITY DETERMINATIONS WITH DIFFERENT TYPES OF COAL.

The pressures produced by the combustion of different dusts varies from zero to 10 pounds per square inch, depending on the chemical and physical properties of the dust and the temperature of the platinum wire. The results obtained with four typical coals are represented graphically in figure 4; and the results of analyses of the coals are given in the following table·

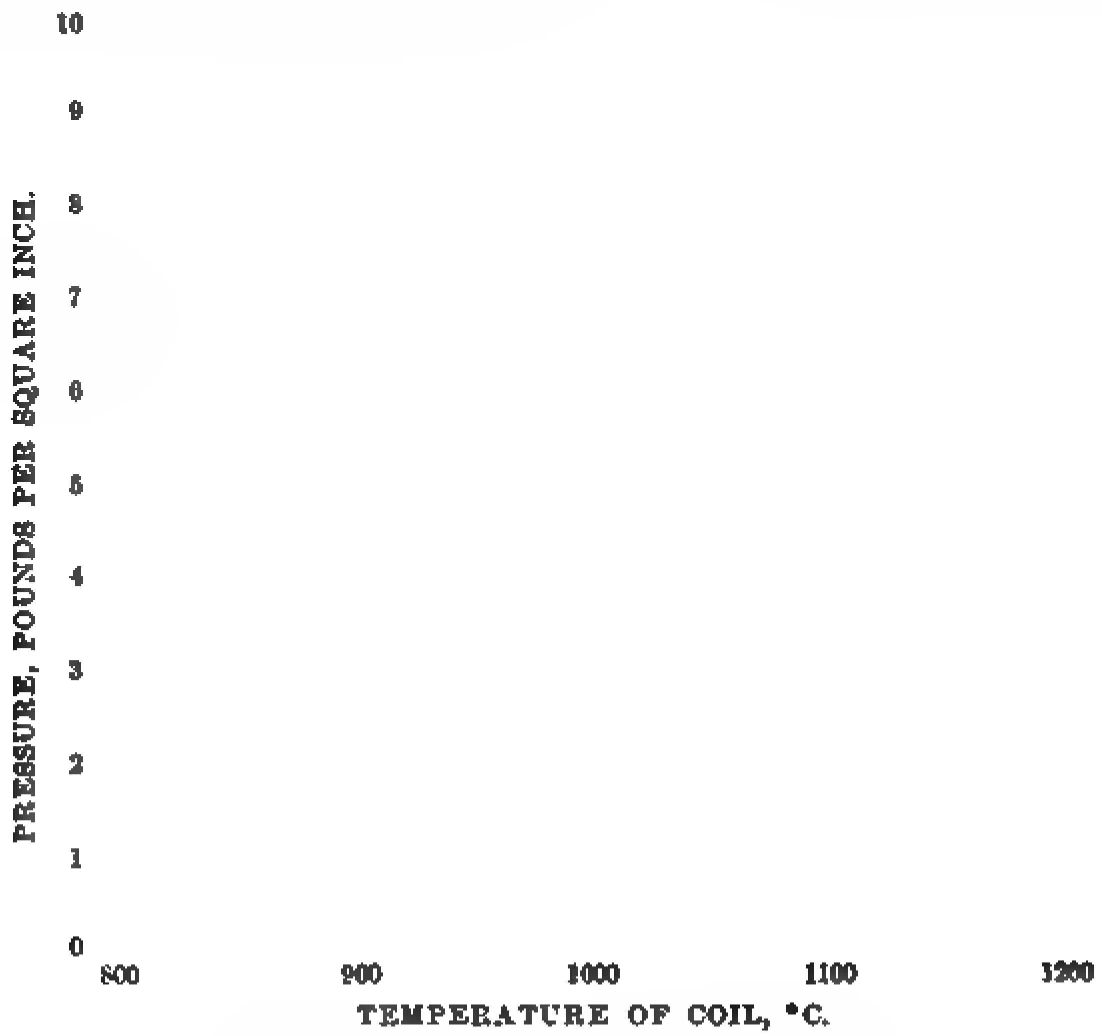

FIGURE 4.—Inflammability curves for four typical coals.

Results of analyses of samples of four typical coals.

Kind of coal.	Laboratory No.	Moisture.	Volatile matter	Fixed carbon.	Ash.	Sulphur.	Hydrogen.	Carbon	Nitrogen.	Oxygen.	Calorific value.
		P. cent.	*P. cent*	*P. cent.*	*P. cent.*	*P. cent.*	*P. cent.*	*P. cent.*	*P. cent.*	*P. cent.*	*B. t. u.*
Anthracite .	15,000	1.58	6.82	80.40	11.20	0.74	2.70	81.03	1.01	3.32	13,124
Semibituminous...	10,630	.51	18.41	77.40	3.68	.64	4.90	85.69	1.60	3.49	15,052
Bituminous	10,845	1.81	35.76	56.41	6.02	1.20	5.32	77.04	1.47	8.95	13,987
Subbituminous......	10,822	13.94	35.12	45.44	5.50	.56	6.01	58.40	1.08	28.36	10,280

The pressure developed by the combustion of anthracite dust is so low as to be inappreciable even at high temperatures, and there is little if any combustion. The method of testing inflammability used in this investigation is not suitable for testing coals low in volatile matter and high in fixed carbon, but such coals are not found in Illinois. Subbituminous dust is distinguished from bituminous dust

A. APPEARANCE OF FLAME WITH PITTSBURGH STANDARD DUST, TEMPERATURE OF COIL 1,000° C.

B. APPEARANCE OF FLAME WITH PITTSBURGH STANDARD DUST; TEMPERATURE OF COIL 1,100° C.

A. APPEARANCE OF FLAME WITH PITTSBURGH STANDARD DUST; TEMPERATURE OF COIL 1,200° C.

B. APPEARANCE OF FLAME WITH DUST THROUGH 20-MESH AND OVER 40-MESH SCREEN; TEMPERATURE OF COIL 1,200° C.

not only by the higher pressures produced on combustion, but by the high pressures being developed at a much lower temperature. There are no subbituminous coals in Illinois. The ratio of volatile matter to fixed carbon of Illinois coals does not vary widely, so that the method of testing probably shows closely the relative inflammability when the ash and the moisture contents are not high.

INFLUENCE OF VARIOUS FACTORS ON INFLAMMABILITY CURVE.

The curve for the bituminous dust exhibits a characteristic common to all of the Illinois coals tested and to a large number of bituminous coals. With rise in temperature the pressure increases slowly at first up to a certain temperature, usually between 1000° and 1100° C., where the curve bends sharply upward and the pressure rises rapidly. After a small rise in temperature beyond this point the curve bends sharply to the right, and the increase in pressure with further rise in temperature is slight. The temperature at which the pressure curve

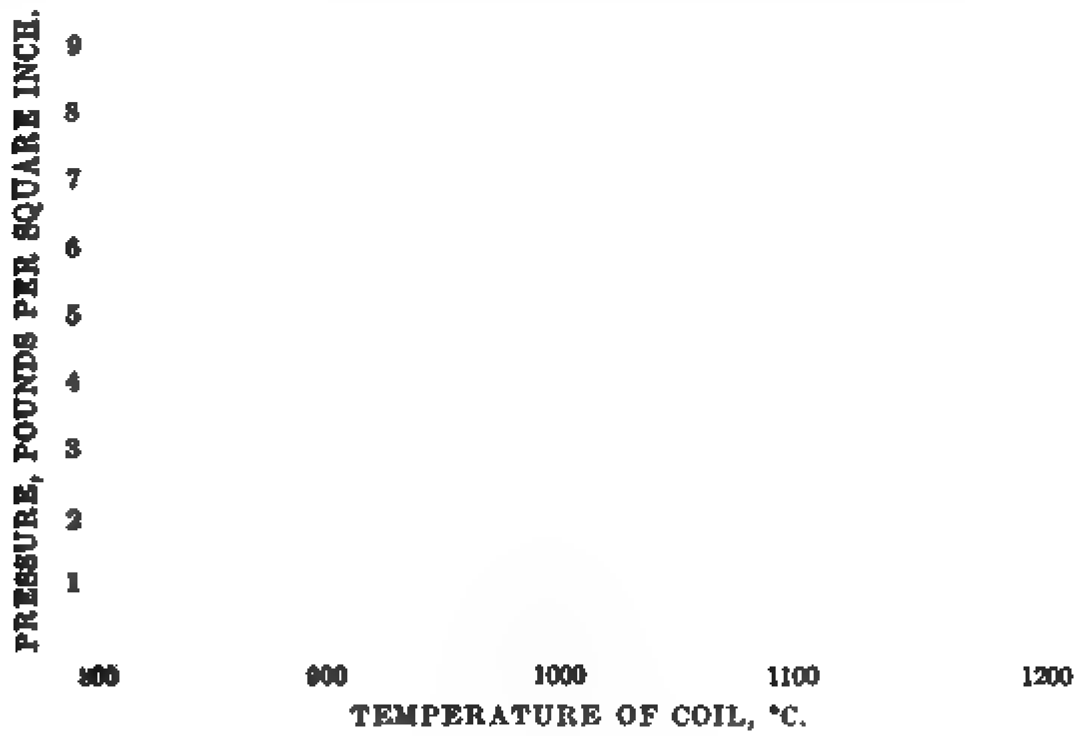

FIGURE. 5.—Inflammability curve for Pittsburgh standard dust with pressures determined at intervals of 20° between 1000° and 1100° C

bends upward has in this report been termed by the authors the "critical temperature."

The exact shape of the inflammability curve for Pittsburgh standard dust in the vicinity of the critical temperature was determined by making tests at intervals of 20° between temperatures of 1000° and 1100° C. The results are represented graphically in figure 5.

The critical temperature seemingly marks the dividing line between combustion in the immediate vicinity of the coil at lower temperatures and a flame that is propagated throughout the dust cloud at higher temperatures. The difference in the character of the flame obtained with Pittsburgh standard dust at temperatures below and above the critical temperature is shown by figures Plate II, *A* and *B*,

and Plate III, *A*. Reference has been made (p. 17) to the practical value of the critical temperature of Pittsburgh standard dust as a point of reference for controlling the temperature of the platinum coil.

As previously stated, all the inflammability tests of face samples were made on samples ground to pass through a 200-mesh screen. The reason for adopting this practice is that the finer the dust the greater is its inflammability. The influence of the size of the dust particles on the phenomena taking place in the flask is shown by Plates III, *B*, IV, *A*, and *B*, and V, *A* and *B*, which are reproduced from photographs of the flames produced by various sized Illinois dusts.

The flame is propagated throughout the dust cloud only with dusts fine enough to pass a 200-mesh screen.

MEASURE OF RELATIVE INFLAMMABILITY.

The inflammability curves in figure 4 indicate that the relative inflammability of the dusts tested by the method used in these experiments may be expressed in terms of any one of several variables; for example, the area below the inflammability curves, the temperature required to develop a certain pressure, or the pressure developed by a certain temperature. All three methods of expressing the results were applied to the data obtained in the study of the face samples of Illinois coal, and the order of inflammability was practically the same with each method. The last-named variable, namely, the pressure developed at a certain coil temperature (1200° C.), has been used by the authors as a measure of relative inflammability.

CLASSIFICATION OF COAL-MINE DUSTS.

Coal-mine dusts may be classified with respect to their inflammability into three groups, as follows: (1) Dusts that, when in suspension in air, may be ignited by a flame from a blown-out shot or by the flame of an oil lamp and give rise to an explosion; (2) dusts that are not ignited by the flame from a blown-out shot or by the flame of an oil lamp, but will propagate an explosion that has originated in a more inflammable mixture of dust or of gas or of both; and (3) dusts that will not propagate an explosion.

Laboratory tests of inflammability furnish a measure of the relative inflammability of dusts, and from the results of such tests dusts may be arranged in the order of their inflammability. However, as the conditions in the laboratory differ so widely from the conditions in the mine, the results of laboratory tests alone do not furnish reliable information as to whether a given dust will propagate a mine explosion. The laboratory results must first be carefully compared with the results of tests on a large scale and under conditions such as exist in

A. APPEARANCE OF FLAME WITH DUST THROUGH 60-MESH AND OVER 80-MESH SCREEN; TEMPERATURE OF COIL 1,200° C.

B. APPEARANCE OF FLAME WITH DUST THROUGH 100-MESH AND OVER 120-MESH SCREEN; TEMPERATURE OF COIL 1,200 C.°

A. APPEARANCE OF FLAME WITH DUST THROUGH 170-MESH AND OVER 200-MESH SCREEN; TEMPERATURE OF COIL 1,200° C.

B. APPEARANCE OF FLAME WITH DUST THROUGH 200-MESH SCREEN; TEMPERATURE OF COIL 1,200° C.

a mine. Only after such a comparison has been made is it possible to predict the behavior of dusts in the mine from the results of laboratory tests.

One of the important problems investigated at the experimental mine of the Bureau of Mines was the determination of the limits between the several classes of dusts named above. At the time the investigation of the inflammability of Illinois dusts was undertaken, the investigations at the experimental mine were in the preliminary stage and no progress had been made on this problem. The laboratory method used in the experiments described in this report was intended especially to furnish information concerning the relative inflammability of dusts.

Since the completion of the experimental part of the present investigation the limits between the three important classes of dusts have been established at the experimental mine. It has been shown that dusts that develop a pressure of 0.5 pound or more in the laboratory apparatus used in this investigation may be ignited by the flame from a blown-out shot, and that certain dusts that develop no measurable pressure in the laboratory test will propagate an explosion. The laboratory method as used in the tests of Illinois dusts is, therefore, not sensitive enough to distinguish between dusts belonging to group 2 and those belonging to group 3.

By recent modifications in the laboratory method measurable pressures are obtained with all dusts that are capable of propagating explosions in the mine.

INFLAMMABILITY OF THE FACE SAMPLES.

TABULATED DATA.

Analyses and inflammability tests were made of 95 face samples from 94 mines. The results are presented in Table 1. The samples are arranged in descending order of inflammability as measured by the pressure developed at a coil temperature of 1200° C. At that temperature the range of pressure is 4.9 to 10.1 pounds per square inch. Dust developing a 0.5-pound pressure or more in the laboratory apparatus, when thrown into suspension in a mine, may be ignited by the flame from a blown-out shot and give rise to an explosion. Coal from any one of the 94 Illinois mines studied, if ground to the proper degree of fineness, will yield a highly inflammable dust, and one that, under suitable conditions, may be expected to give rise to a violent explosion.

TABLE 1.—*Results of analyses and of inflammability tests of face samples.*

												Pressure at a coil temperature of—				
												0° C.	900° C.	1,000° C.	1,100° C.	1,200° C.
												unds per uare nch.	*Pounds per square inch.*	*Pounds per square inch.*	*Pounds per square inch.*	*Pounds per square inch.*
												0.3	1.9	4.2	9.3	10.1
												.9	2.3	4.4	8.6	9.8
												.3	1.6	3.0	8.3	9.1
												.9	1.7	5.1	8.1	8.9
												.7	1.7	5.1	8.4	8.9
												.3	2.4	7.2	8.2	8.9
												0	.3	3.3	8.0	8.7
												.2	1.9	3.5	8.0	8.7
												0	1.5	2.3	8.2	8.7
												0	1.2	1.9	8.0	8.7
												0	.4	2.6	7.9	8.7
												0	1.2	2.6	7.7	8.6
												0	.9	2.4	7.7	8.4
												0	.5	2.3	7.7	8.4
												.9	2.1	6.8	7.9	8.4
												.3	2.1	3.5	7.9	8.4
												.5	2.3	6.1	7.7	8.4
												0	1.0	2.4	7.0	8.4
												0	1.4	2.8	7.5	8.2
												.2	.5	2.8	7.5	8.2
												.3	.9	1.9	7.5	8.2
												.5	1.2	2.1	7.3	8.2
93	13.27	5.39	38.67	46.61	9.33	2.41	12,221	5.16	13.78	67.90	1.53	.5	1.7	4.4	8.0	8.2
100	7.05	3.55	42.30	43.30	10.85	3.32	12,351					.5	2.4	4.2	7.5	8.2
4	11.22	5.67	39.21	47.93	7.19	2.73	12,539	5.38	13.65	69.78	1.27	0	1.0	2.6	7.2	8.1
52	4.55	2.97	39.13	46.69	11.21	3.26	12,245					.3	1.7	3.2	7.2	8.1
5	9.92	6.77	40.63	45.59	7.01	2.92	12,240	5.41	16.43	68.00	1.28	0	1.2	2.8	7.2	8.0
7	9.57	7.24	40.09	47.02	5.65	2.28	12,433	5.49	15.80	69.49	1.29	0	1.6	2.6	7.2	8.0
19	10.20	4.15	42.30	42.22	11.33	4.52	11,957					.5	1.9	3.1	7.0	8.0
66	6.86	7.90	40.20	41.03	10.87	4.72	11,309	5.25	15.89	62.20	1.07	.2	1.0	2.3	6.6	8.0
80	5.26	5.24	38.61	43.74	12.41	3.99	11,533	4.97	12.73	64.72	1.18	.2	.7	1.7	7.2	8.0
92	13.74	4.42	36.72	46.51	12.36	2.46	11,929	4.77	11.57	67.41	1.44	.7	1.2	1.9	7.3	8.0
Extra	6.91	2.75	38.66	46.03	12.64	4.10	12,317					.5	1.7	4.5	7.0	7.9
11	9.10	4.52	39.68	46.18	9.67	3.35	12,373	5.20	12.16	68.41	1.21	.2	1.6	3.8	7.8	7.9
34	13.80	4.83	38.92	46.15	10.10	3.64	11,886	5.07	13.90	66.04	1.25	.3	.9	1.7	7.2	7.9
64	4.16	5.31	37.26	48.70	8.73	2.26	12,290	5.05	13.72	68.60	1.42	.2	.9	3.7	6.8	7.9

70	6.50	6.44	39.34	42.07	12.15	4.64	11,374	5.00	14.77	62.37	1.07	.2	.9	2.0	7.2	7.9
81	4.80	6.64	41.40	41.33	10.63	4.14	11,637	5.22	15.22	63.96	.83	0	.3	1.7	7.0	7.9
38	7.45	7.58	39.05	42.33	11.04	4.10	11,315	5.08	15.50	63.03	1.25	0	.5	1.7	7.3	7.8
36	11.83	4.81	39.19	44.44	11.56	3.94	11,590	4.95	14.17	64.10	1.28	.2	.7	1.6	6.8	7.7
43	4.54	3.57	36.48	51.30	8.65	2.20	12,778					.3	.7	1.9	7.0	7.7
75	9.00	5.94	40.66	43.24	10.16	3.89	11,691	5.38	14.09	65.34	1.14	0	.5	1.4	6.5	7.7
29	11.50	5.66	38.74	43.62	11.98	3.09	11,500	5.20	13.98	64.57	1.18	0	.5	1.9	6.6	7.5
30	12.70	4.45	39.09	43.37	12.89	3.11	11,633	5.18	12.61	65.05	1.16	0	.5	1.6	6.3	7.5
32	10.79	4.55	40.04	42.21	13.20	3.55	11,749	5.07	11.47	65.55	1.16	0	.4	1.6	6.5	7.5
37	6.27	8.08	38.35	42.14	11.43	4.11	11,090	5.14	15.72	62.37	1.23	0	.3	1.3	6.8	7.5
25	10.75	4.76	40.03	42.78	12.43	3.74	11,723	5.01	11.65	65.95	1.22	.7	1.7	2.4	6.3	7.5
40	10.65	5.44	39.11	44.13	11.32	4.03	11,623	5.10	13.55	64.74	1.26	.1	.9	1.8	6.6	7.4
31	11.86	5.85	39.65	42.64	11.86	3.41	11,574					0	.3	1.7	6.5	7.3
44	4.01	3.28	36.43	51.48	8.81	2.69	12,691	5.19	10.72	70.97	1.02	.3	.9	1.7	6.8	7.3
67	8.12	7.16	39.81	42.11	10.92	4.11	11,381	5.27	15.40	63.16	1.14	.2	.7	2.1	5.9	7.3
82	7.27	5.28	41.12	42.00	11.60	4.55	11,700	5.12	14.00	63.64	1.09	.1	.7	1.2	6.6	7.3
86	5.65	5.80	38.46	43.75	11.99	3.97	11,552	5.01	13.81	64.02	1.20	0	.7	1.4	6.3	7.2
39	6.98	7.11	38.07	41.53	13.29	4.70	11,029	4.95	14.94	60.91	1.21	0	.5	1.9	6.5	7.2
45	4.31	3.41	36.17	52.12	8.30	2.51	12,706	5.24	11.84	70.50	1.61	.2	.7	1.6	6.6	7.2
48	5.05	3.81	35.50	51.64	9.05	2.41	12,575	5.11	11.09	70.79	1.55	0	.5	1.4	6.5	7.2
84	5.85	7.77	38.48	43.08	10.67	4.35	11,324	5.18	15.59	63.14	1.07	0	.5	1.4	6.3	7.2
88	4.42	5.83	37.42	44.06	12.69	3.76	11,317	4.92	13.95	63.55	1.13	0	.7	1.7	6.5	7.2
49	2.90	2.77	38.50	47.33	11.40	4.60	12,343	5.05	9.96	67.54	1.45	0	.3	1.2	6.6	7.2
21	6.87	5.43	40.02	46.31	8.24	2.72	12,388	5.36	13.33	68.95	1.40	.3	1.4	2.8	5.9	7.0
22		4.03	36.90	49.82	9.25	2.42	12,452	5.18	12.54	69.24	1.37	.3	1.4	2.6	6.3	7.0
83	6.35	5.81	38.94	43.13	12.12	4.33	11,399	4.95	14.20	63.49	.91	0	.3	1.4	6.1	7.0
71	4.48	8.01	40.23	40.80	10.96	4.07	11,322	5.22	16.36	62.37	1.02	0	.5	1.6	5.8	6.9
76	8.55	6.22	39.94	42.15	11.69	3.99	11,534	5.04	14.17	63.96	1.15	0	.3	1.4	5.9	6.8
87	4.33	6.66	35.67	46.09	11.58	3.99	11,466	4.99	14.56	63.67	1.21	0	.3	1.4	5.9	6.8
90	5.44	6.40	36.50	46.22	10.88	3.55	11,479					0	.2	1.6	5.4	6.8
73	7.22	7.93	40.74	42.25	9.08	3.93	11,527	5.36	16.48	64.00	1.15	0	.5	1.6	5.9	6.8
58	6.12	4.02	37.16	50.47	8.35	.92	12,483					0	.2	.5	5.9	6.7
15	4.37	4.35	35.20	54.96	5.49	1.52	13,115					0	.7	1.9	5.9	6.6
47	3.40	2.65	36.15	48.38	12.82	3.40	12,263	4.84	9.73	67.87	1.34	0	.7	1.0	5.9	6.6
69	8.10	6.71	39.33	41.59	12.37	4.52	11,232	5.33	14.33	62.32	1.13	0	.5	1.7	5.2	6.6
55	6.38	3.48	36.37	49.25	10.90	2.83	12,154					0	.2	.9	5.6	6.5
46	4.34	4.35	35.40	50.91	9.34	2.84	12,411	5.14	11.26	69.89	1.53	0	.5	1.2	5.8	6.3
50	3.72	6.41	34.47	51.18	7.94	1.04	12,253	5.34	14.90	69.14	1.64	.2	.7	1.9	4.9	6.3
89	4.99	7.91	35.33	47.53	9.23	2.04	11,723	5.12	16.06	66.16	1.39	0	.5	1.4	5.4	6.3
42	7.14	7.69	37.99	44.73	9.59	3.69	11,497	5.19	16.80	63.51	1.22	0	.5	1.6	5.2	6.1
16	3.27	5.77	34.51	53.68	6.04	1.36	12,856	5.25	13.07	72.74	1.54	0	.5	1.4	5.2	5.9
41	9.39	6.63	37.20	43.54	12.63	3.42	11,214	4.98	14.71	63.06	1.22	.2	1.0	1.7	5.2	5.9
72	4.75	8.95	40.18	40.37	10.50	4.26	11,164	5.26	17.09	61.72	1.17	0	.5	1.2	4.7	5.9
85	4.67	8.43	37.53	43.73	10.31	3.59	11,313	5.29	a 16.60	63.02	1.19	0	.3	1.4	4.9	5.9
56	3.14	5.31	35.37	48.55	10.77	2.53	11,974	5.17	a 12.56	67.57	1.40	0	.2	1.3	5.1	5.9
14	3.88	5.07	34.33	53.34	7.26	1.58	12,717	5.10	12.62	71.89	1.55	0	.7	1.2	4.9	5.8
51	6.51	4.44	35.24	52.62	7.70	1.27	12,555	5.20	13.75	70.48	1.60	0	.2	.3	5.1	5.8
74	6.57	8.44	39.61	41.12	10.83	4.80	11,122	5.29	16.48	61.46	1.14	0	.3	1.0	4.5	5.8
63	5.07	5.39	32.91	52.25	9.45	1.67	12,222	4.95	13.04	69.37	1.52	0	.4	1.0	4.2	5.7

a Includes nitrogen content.

TABLE 1.—*Results of analyses and of inflammability tests of face samples*—Continued.

Mine No.	Air-drying loss.	Proximate analysis.					Calorific value.	Ultimate analysis.				Pressure at a coil temperature of—				
		Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.		Hydrogen.	Oxygen.	Total carbon.	Nitrogen.	800° C.	900° C.	1,000° C.	1,100° C.	1,200° C.
		Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*B. t. u.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Pounds per square inch.*	*Pounds per square inch.*	*Pounds per square inch.*	*Pounds per square inch.*	*Pounds per square inch.*
61	3.11	6.82	34.88	50.95	7.35	1.21	12,259	5.14	15.57	69.21	1.52	.1	.5	1.6	4.0	5.6
12	6.60	3.72	33.53	55.67	7.08	1.21	12,906	....	....	....	1.63	.2	.3	.9	4.4	5.4
13	6.42	4.32	33.68	56.11	5.89	1.07	13,075	5.13	12.00	74.42	1.49	0	.3	1.0	4.2	5.4
54	7.51	4.28	37.77	47.11	10.84	.95	11,934	....	....	....	....	0	.2	.5	4.3	5.4
60	5.48	3.87	35.37	51.24	9.52	2.22	12,346	4.92	11.75	70.13	1.46	0	.2	.7	4.1	5.4
65	6.29	4.18	35.09	52.00	8.73	1.79	12,478	5.05	12.17	70.78	1.48	0	.3	.9	4.3	5.4
53	6.68	4.27	35.24	51.48	9.01	.72	12,280	....	....	....	....	.3	.5	.9	4.2	5.1
57	3.45	7.05	32.38	51.51	9.06	.67	12,049	4.99	15.15	68.52	1.61	0	.3	1.0	4.2	5.1
59	3.91	7.24	33.03	50.22	9.51	1.47	11,902	5.08	14.83	67.59	1.52	0	.3	1.4	3.1	4.9
62	3.01	6.79	33.86	51.27	8.08	1.12	12,159	5.12	15.24	68.84	1.60	0	.2	1.6	4.0	4.9

INFLAMMABILITY CURVES.

Inflammability curves for a number of the face samples are contained in figures 6, 7, and 8, in which the abscissa is the temperature of the platinum coil and the ordinate is the pressure developed in the flask. The curves in figure 6 represent the most inflammable dust from each district and those in figure 7 the least inflammable. Curves for all face samples from district 7 are shown in figure 8.

INFLAMMABILITY OF RIB DUSTS.

The study of the face samples has shown that all Illinois coals, when ground to a sufficient degree of fineness, yield dusts that are highly

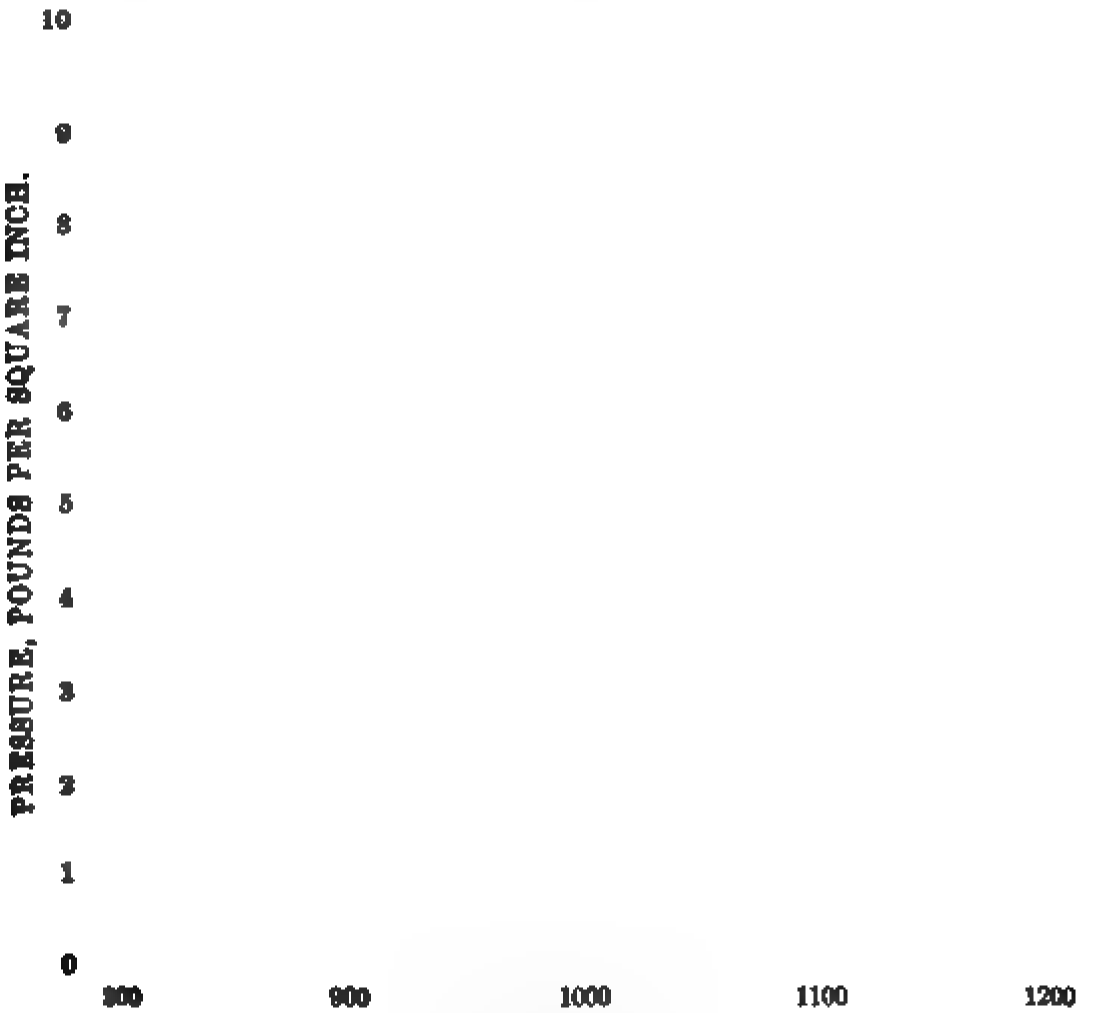

FIGURE 6.—Inflammability curves for the most inflammable face sample from each district. Numbers on curves refer to districts shown on figure 1.

inflammable. The purpose of studying the rib and the road dusts is to ascertain to what extent inflammable dusts are formed and are present in the mines.

TABULATED DATA.

Table 2 contains the results obtained in the study of rib dusts. The samples are arranged in the order of the inflammability of the portion ground to pass a 200-mesh screen, as measured by the pressure developed at a coil temperature of 1,200° C. The table follows:

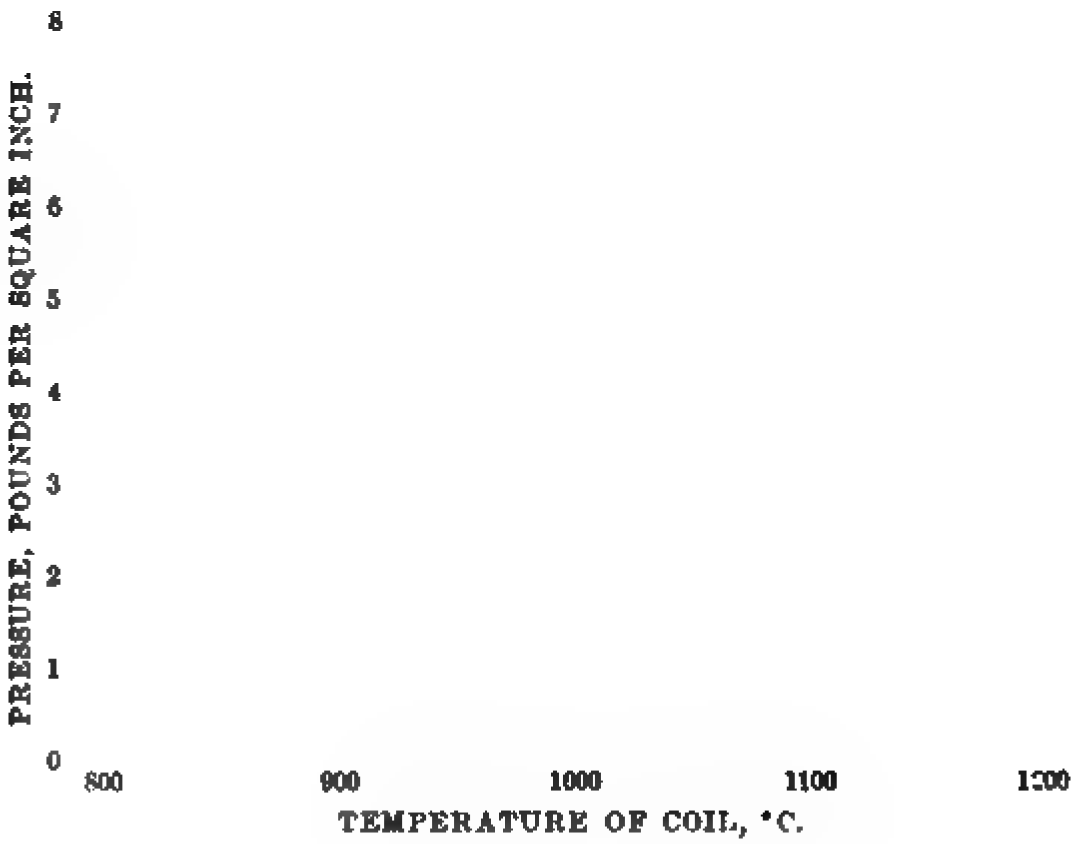

FIGURE 7.—Inflammability curves for the least inflammable face sample from each district. Numbers on curves refer to districts shown on figure 1.

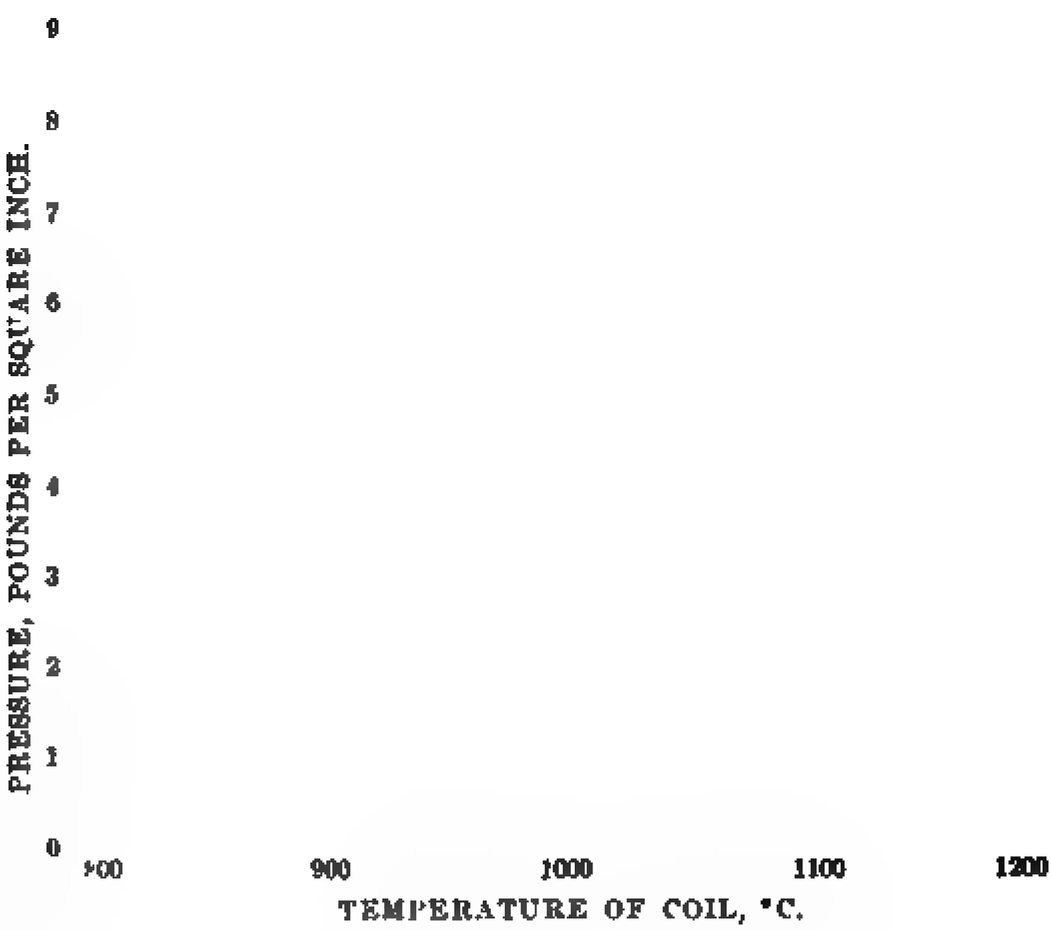

FIGURE 8.—Inflammability curves for all face samples of district 7. Numbers on curves refer to mine numbers shown in Table 1.

TABLE 2.—*Results of tests of rib dusts.*

					R… st igh D- th e.						mate analy Fixed carbon.							' square inch of— nd dust through a 200-mesh e at a cell temperature of— 900° C.	1,000° C.	1,100° C.	1,200° C.
											12							19	20	21	22
		[a]	*P. ct.*	*Ounces.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*
72......	1	0	15.79	0.016	29.37	13.26	6.87	6.27	3.70	36.50	44.45	15.35	6.84	11,007	0.0	5.6	0.0	0.3	1.9	8.6	9.3
26......	1	5	2.96	.162	86.17	56.59	41.11	24.73	3.20	37.70	38.32	20.73	3.89	10,651	.7	6.5	.0	1.0	2.4	8.4	9.1
9......	1	2	20.00	.330	38.53	17.24	10.92	7.73	2.38	41.00	38.82	17.80	3.69	11,014	.0	7.0	.0	1.0	2.6	8.4	9.1
1......	1	2	7.55	.314	31.92	15.06	9.37	5.47	2.10	39.80	36.70	21.40	4.22	10,406	.0	5.2	.5	1.2	2.0	7.7	8.9
3......	1	2	17.83	.195	39.67	18.86	12.88	7.96	2.10	40.60	34.90	22.40	5.30	10,771	.2	7.9	.0	.3	.9	7.9	8.8
28......	1	5	23.23	.170	62.94	33.54	21.96	12.50	3.00	39.20	40.55	17.25	4.27	10,823	.6	7.3	.2	.9	2.1	8.2	8.7
31......	1	5	19.21	.655	42.04	16.68	9.28	5.53	2.78	40.17	39.45	17.00	2.54	10,624	.2	6.5	.5	1.9	3.8	7.9	8.7
66......	1	6	17.39	.221	57.38	28.77	19.27	11.47	4.45	38.85	45.27	11.43	4.46	11,542	.7	7.7	.7	1.6	2.3	8.4	8.7
71......	1	6	7.91	.672	16.19	6.78	4.35	2.86	3.13	41.22	39.25	16.40	3.69	11,221	.3	4.5	.3	1.0	2.4	2.0	8.7
29......	1	5	21.63	.352	52.20	26.58	18.42	11.43	2.43	39.07	39.17	19.33	4.17	10,939	1.7	7.2	.2	.9	2.3	8.0	8.6
32......	1	5	2.12	.179	59.63	27.93	18.92	11.60	2.10	38.08	40.12	19.70	5.65	11,147	.9	8.9	.6	1.4	2.1	2.1	8.6
36......	2	5	22.45	.318	50.51	26.27	17.50	10.83	4.33	37.35	41.39	16.93	4.57	11,006	.5	6.5	.3	.9	2.2	7.7	8.6
72......	2	6	15.81	.027	40.71	16.24	9.91	5.86	5.23	38.62	41.15	16.80	4.73	10,262	.3	3.3	.0	.3	.9	7.7	8.4
77......	1	6	7.13	.946	67.98	42.79	30.83	20.15	6.15	37.90	43.63	12.32	4.57	11,045	.3	6.6	.2	1.2	2.0	6.6	8.4
25......	1	5	21.64	.750	61.04	31.51	21.59	13.15	3.35	37.10	43.90	15.65	4.33	11,428	.2	7.2	.3	.9	2.6	7.9	8.4
4......	1	2	15.06	....	17.90	5.88	3.26	1.80	....	....	....	....	....	....	.0	4.2	.0	.3	1.9	7.7	8.2
23......	2	5	21.29	.088	69.07	40.07	26.67	16.00	4.23	33.50	40.69	21.58	5.26	10,121	.3	6.8	.0	.5	2.3	7.5	8.2
44......	2	5	16.35	.398	72.81	39.31	27.29	16.14	3.00	32.30	52.32	11.78	4.40	11,923	.7	6.1	.2	.9	2.4	7.6	8.2
38......	1	5	19.49	.226	56.09	31.82	22.24	13.50	4.78	38.50	39.08	17.64	5.42	10,706	.7	6.3	.5	.9	1.6	7.2	8.0
74......	1	6	20.89	.147	38.15	18.71	12.62	8.23	3.65	38.90	40.97	16.48	4.87	10,870	.2	7.5	.0	.5	2.6	7.3	8.0
19......	1	1	19.77	......	60.86	35.39	22.44	12.17	2.80	36.90	39.35	20.95	7.33	10,667	.5	6.8	.5	1.7	3.0	7.3	8.0
27......	1	5	19.65	.375	49.20	20.81	11.98	7.07	2.75	38.57	36.95	21.73	3.71	10,530	.0	5.9	.2	.7	1.9	7.5	7.9
70......	1	6	15.36	.160	32.89	12.85	7.74	4.59	3.00	40.76	34.20	22.05	5.30	10,204	.3	6.8	.2	.7	1.6	7.0	7.9
81......	1	6	25.49	.227	66.80	55.30	42.30	26.39	3.23	35.62	42.42	18.73	4.20	10,766	.7	5.4	.0	.7	1.6	7.2	7.9
68......	1	6	18.80	.120	55.28	21.69	12.22	7.66	3.48	34.97	45.90	15.65	4.33	11,097	.2	1.6	.2	.3	1.6	7.3	7.9
31......	2	5	8.72	.066	72.73	51.54	40.36	27.07	3.78	37.52	35.17	23.53	5.23	9,547	.5	5.8	.0	.3	1.7	7.5	7.9
34......	1	5	24.49	.245	53.99	25.18	15.76	9.46	3.75	37.18	42.77	16.30	5.06	11,110	.2	5.8	.2	1.1	1.7	7.7	7.9

[a] Figure 1 indicates that sample was taken from a room; figure 2, that sample was taken from the secondary haulageway; and figure 3, that sample was taken [redacted] the main haulageway.

TABLE 2.—*Results of tests of rib dusts*—Continued.

Mine No.	Place of sampling.	Seam.	Air-drying loss.	Dust per square foot.	Results of sizing tests.				Proximate analysis.					Calorific value.	Pressure per square inch of—						
					Dust through a 20-mesh sieve.	Dust through a 60-mesh sieve.	Dust through a 100-mesh sieve.	Dust through a 200-mesh sieve.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.		Dust through a 20-mesh sieve at a coil temperature of 1,200° C.	Dust through a 200-mesh sieve at a coil temperature of 1,200° C.	Ground dust through a 200-mesh sieve at a coil temperature of— 800° C.	900° C.	1,000° C.	1,100° C.	1,200° C.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
			P. ct.	*Ounces.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*
15......	2	2	18.70	.106	52.88	28.66	19.76	12.12	3.50	32.35	46.40	17.75	2.71	10,870	.2	4.4	.0	.8	4.4	7.1	7.8
73......	2	6	14.09	.183	30.40	18.78	14.49	10.22	4.25	33.75	42.45	19.55	5.00	10,253	.3	4.7	.0	.3	1.9	7.0	7.7
78......	1	6	21.91	.220	35.25	15.74	9.94	5.42	3.63	34.57	36.67	25.13	4.80	9,508	.0	4.6	.0	.9	2.1	6.8	7.7
27......	2	5	23.01	.180	53.37	15.59	10.03	6.57	4.60	37.15	37.15	21.10	4.87	10,017	.2	5.9	.0	.7	2.1	7.2	7.7
17......	1	1	21.15	....	50.06	14.26	7.28	3.13	3.45	37.83	34.02	24.70	7.53	9,887	.2	5.8	.5	1.9	3.3	6.8	7.7
18......	1	1	21.36	....	71.96	38.20	22.66	11.33	3.00	36.85	36.61	23.54	8.86	10,184	.2	7.7	.3	2.1	3.5	6.6	7.5
97......	2	7	45.39	.111	....	47.91	22.54	10.73	....	....	....	....	....	....	....	....	.0	.3	1.0	3.3	7.5
82......	1	6	17.48	.151	40.53	18.04	12.72	7.81	2.73	37.70	34.87	24.70	4.58	9,850	.3	1.2	.0	.7	1.2	7.0	7.5
44......	1	5	15.20	.106	51.83	24.45	18.04	11.74	2.80	34.90	42.12	20.48	4.31	10,879	.5	3.7	.2	.7	1.6	5.9	7.5
15......	1	2	15.89	.104	41.60	19.32	13.98	8.99	2.43	32.52	48.31	16.74	4.56	11,453	.0	4.3	.0	.2	.9	5.1	7.4
39......	1	5	14.79	.299	77.05	49.36	30.40	15.52	7.80	37.50	37.63	17.07	4.56	10,442	.7	6.6	.2	.8	1.2	5.8	7.4
24......	1	1	25.36	....	80.18	43.54	30.28	16.63	2.55	37.00	38.10	22.35	6.26	10,710	.3	7.2	.3	1.4	2.6	6.5	7.3
5.......	1	3	18.85	.165	31.01	18.53	9.95	7.25	....	....	....	....	....	....	.2	1.7	.0	.2	1.7	6.1	7.3
12......	1	2	13.04	.390	29.53	15.05	8.56	5.20	2.48	32.82	45.10	19.60	2.79	10,969	.2	4.4	.0	.2	.9	5.8	7.3
36......	1	5	22.09	.041	65.26	43.08	31.26	18.92	5.40	35.95	40.40	18.25	4.40	10,570	.7	4.5	.2	.9	2.4	6.6	7.3
78......	2	6	18.53	.199	48.56	21.49	15.37	9.61	3.30	33.05	44.72	18.93	4.02	10,672	.0	.7	.0	.5	1.7	6.8	7.3
48......	1	5	13.45	.129	61.46	33.30	24.69	16.31	2.63	32.07	53.70	11.60	3.12	12,139	.5	2.1	.0	.3	1.4	6.4	7.2
73......	1	6	11.19	.182	15.99	5.34	3.57	2.44	....	....	....	....	....	....	.2	.3	.0	.4	1.1	5.4	7.2
13......	1	2	14.30	.137	53.92	32.13	22.70	13.02	2.78	32.80	52.52	11.90	2.54	12,325	.2	5.6	.0	.7	1.4	6.5	7.1
13......	2	2	14.79	.043	53.18	20.90	14.46	8.16	4.05	32.15	50.47	13.33	3.01	11,309	.2	2.3	.0	.3	1.0	4.4	7.1
47......	1	5	14.73	.085	61.70	33.29	22.86	13.56	1.80	34.18	44.47	19.25	4.81	11,196	.0	5.9	.0	.4	.9	6.5	7.0
88......	3	6	....	....	....	....	....	....	4.33	33.12	38.27	24.28	3.72	9,666	.3	4.2	.0	.3	1.3	4.7	7.0
37......	1	5	20.98	.292	79.67	50.08	35.97	18.56	4.88	35.40	40.39	19.33	5.72	10,546	.3	5.4	.0	.2	.5	5.6	7.0
69......	1	6	23.48	.193	76.29	43.09	31.14	17.67	3.55	34.48	42.02	19.95	4.27	10,568	.3	4.2	.0	.5	1.6	6.1	6.8
75......	1	6	17.43	.738	65.73	39.30	26.46	16.39	4.85	35.63	41.69	17.83	4.81	10,508	.3	5.1	.2	.3	1.9	5.4	6.8
83......	2	6	13.23	.070	26.73	11.41	7.56	4.35	3.05	37.40	36.27	23.28	4.40	9,896	.0	4.2	.0	.3	1.4	6.3	6.8
79......	1	6	16.40	.085	40.81	22.70	16.65	10.23	2.95	34.62	40.98	21.45	3.62	10,283	.0	.7	.0	.2	1.6	6.3	6.8
91......	1	6	14.28	....	42.33	23.13	14.20	8.23	4.33	33.97	48.22	13.48	1.27	11,543	.5	1.7	.2	.2	1.7	4.9	6.8
67......	1	6	19.90	.094	49.14	16.97	10.52	6.60	3.95	33.80	41.92	20.33	4.23	10,375	.0	1.2	.0	.5	2.3	6.1	6.7
8.......	1	2	18.09	....	46.07	19.18	10.67	5.97	2.65	34.23	34.99	28.13	3.51	9,641	.0	4.9	.0	.3	1.0	5.8	6.7

48	2	5	17.37	.008	44.78	13.06	8.87	5.99	3.55	20.40	45.27	21.78	2.14	9,734	.2	.3	.0	.2	.5	1.2	6.6
52	2	6	3.60	.041	50.37	37.21	30.29	23.29	3.19	38.61	39.70	18.50	3.17	11,018	...	...	.0	.5	.9	4.5	6.6
76	1	6	12.99	.386	39.43	19.65	12.70	7.31	4.10	35.78	40.69	19.43	4.04	10,814	.2	4.5	.0	.3	.7	5.8	6.6
76	2	6	17.38	.168	50.13	25.66	18.86	11.22	5.73	33.95	43.19	18.13	4.29	10,828	.0	1.2	.0	.5	1.7	5.7	6.6
79	2	6	16.60	.139	36.69	17.40	11.70	7.10	3.58	36.57	37.63	22.22	4.24	9,983	.5	5.1	.0	.2	1.4	5.2	6.6
95	1	6	15.41	...	...	63.34	53.20	37.32	...	...	...	...	...	...	...	...	...	.3	.9	3.5	6.6
36	2	5	12.08	.472	87.14	71.83	59.61	40.98	5.80	37.13	29.94	27.13	6.43	8,962	1.4	5.8	.0	.3	2.1	5.6	6.6
29	3	5	9.87	.348	71.06	49.37	33.44	19.12	6.20	37.63	29.66	26.51	6.01	8,924	.3	6.6	.0	.2	.9	2.6	6.6
40	1	5	22.49	.320	64.70	24.11	16.43	6.52	6.00	36.80	39.45	17.76	3.81	10,535	.5	4.2	.0	.5	1.3	4.4	6.6
90	2	6	12.03	.279	39.61	25.29	14.51	7.66	4.55	37.15	41.35	16.95	3.93	10,712	.0	5.8	.2	.3	1.4	5.9	6.6
10	1	2	13.17	...	10.84	2.09	.87	.74	...	...	...	...	...	...	.0	.3	.0	.5	1.9	6.3	6.6
83	1	5	21.72	.284	62.98	17.78	9.89	4.19	5.15	36.40	36.77	21.68	3.51	10,237	.0	4.0	.0	.3	.9	5.8	6.5
100	1	3	4.62	.080	37.46	20.40	14.35	8.53	...	...	...	...	...	...	.2	1.9	.0	.3	.9	5.1	6.5
68	2	6	10.41	.115	76.34	60.97	50.00	40.27	3.03	34.67	38.20	24.10	3.83	10,006	.2	4.7	.2	.7	1.0	5.5	6.5
43	1	5	8.83	.278	55.63	33.22	21.85	12.81	4.28	31.47	46.27	17.98	5.05	10,784	.2	3.8	.0	.2	1.3	4.7	6.3
90	1	6	13.75	.235	68.75	43.56	30.86	16.52	4.35	34.85	40.52	20.28	5.06	10,071	.5	5.8	.0	.2	.7	5.4	6.3
93	1	6	13.95	...	58.02	36.37	29.78	17.64	4.63	37.17	46.62	11.58	1.68	11,774	.2	3.2	.0	.3	1.0	5.8	6.3
86	1	6	19.06	.346	65.31	47.09	36.50	23.84	5.48	34.32	43.65	16.55	5.11	10,640	.3	5.6	0	.3	1.0	5.6	6.2
77	2	6	4.61	.099	66.23	43.27	31.19	16.73	7.40	38.80	31.37	22.43	5.69	9,110	.3	5.1	0	0	.2	4.2	6.1
26	3	5	28.43	.215	84.38	69.54	63.56	55.48	4.00	34.65	26.21	35.14	4.03	7,879	3.5	5.8	0	.8	1.0	5.2	6.1
41	1	5	7.15	.099	31.64	40.13	30.63	19.54	7.28	33.55	40.54	18.63	4.35	10,004	1.0	4.3	0	.4	1.4	5.2	6.1
33	2	5	6.69	.399	88.41	60.22	41.97	24.67	7.50	36.78	36.44	19.28	5.18	9,895	.9	3.6	0	.3	1.7	5.4	5.9
28	3	5	8.83	.019	19.31	6.10	3.55	2.20	...	...	...	...	...	...	0	0	0	.3	1.0	4.9	5.9
62	2	6	24.29	.100	77.73	54.89	48.21	37.59	3.66	33.32	51.21	11.81	2.01	12,103	...	...	0	.5	1.2	4.4	5.9
80	1	6	6.52	.447	87.90	64.13	49.76	32.64	5.65	35.85	38.78	19.78	4.88	9,988	.5	4.2	0	.3	.9	5.2	5.9
52	1	6	8.17	.035	42.51	30.48	24.39	18.09	4.54	35.11	45.46	14.89	1.95	11,426	...	...	0	.2	.9	2.8	5.8
54	2	6	17.72	.065	53.42	28.34	20.82	14.59	5.03	33.16	49.57	12.24	1.06	11,657	...	...	.2	.5	2.3	4.2	5.8
57	2	6	9.76	...	65.30	44.48	36.21	28.44	4.81	30.59	42.58	22.07	2.74	10,123	...	...	0	.5	.9	3.1	5.8
14	1	2	17.64	.295	75.53	43.67	33.00	17.10	2.50	33.38	44.61	19.51	1.59	10,721	.3	4.7	0	.2	.5	4.6	5.8
29	2	5	8.68	.129	52.81	31.40	24.50	19.09	4.50	34.60	26.50	34.40	4.31	7,900	.2	.2	0	.5	1.1	5.1	5.8
95	2	6	24.43	...	...	78.82	70.38	45.01	...	...	...	...	...	...	...	...	0	.9	1.2	3.0	5.8
34	2	5	28.91	.508	64.69	39.35	25.88	14.27	6.45	34.63	34.42	24.50	5.30	8,996	.2	1.7	0	0	.7	1.7	5.7
12	2	2	20.10	...	68.34	34.27	26.72	15.88	2.83	29.77	44.82	22.58	2.19	10,490	0	4.7	0	.2	.7	4.0	5.6
49	1	5	10.04	.176	67.71	48.42	38.12	26.56	...	...	...	...	...	...	0	2.9	0	.2	.7	4.4	5.6
11	1	2	13.01	.225	17.15	5.90	3.91	2.63	...	...	...	...	...	...	0	.2	0	0	.2	4.2	5.6
50	3	6	5.72	.033	38.54	21.73	15.31	9.07	4.33	29.47	41.87	24.33	4.20	9,794	.2	1.2	0	.3	.7	2.1	5.4
63	1	6	16.67	.087	71.98	51.82	41.87	30.74	3.74	33.30	48.46	14.50	1.75	11,569	...	...	0	.3	2.3	5.4	5.4
38	2	5	11.11	.967	81.55	60.76	44.36	24.36	7.38	33.05	38.02	21.55	4.83	9,164	.2	3.1	0	.2	.5	3.1	5.4
65	1	6	19.17	.120	85.09	66.70	51.95	35.99	4.63	32.33	40.20	22.85	3.00	10,022	...	...	0	.3	.5	1.2	5.3
32	2	5	12.35	.615	81.41	57.55	41.66	27.68	5.98	37.62	27.90	28.50	6.62	8,082	.2	3.7	0	.2	.5	3.0	5.3
45	3	5	11.20	.592	73.88	44.68	30.91	17.43	4.60	29.90	44.55	20.95	3.71	10,489	.2	3.1	0	.2	.5	4.2	5.2
46	1	5	14.02	.345	68.03	40.45	24.79	11.12	3.70	30.43	42.30	23.57	5.55	10,138	.9	3.1	0	.5	1.4	4.5	5.1
39	2	5	11.71	.614	89.25	46.41	44.23	22.91	8.85	32.88	30.94	27.33	5.51	8,221	.2	3.1	0	.2	.7	2.5	5.0
71	2	6	7.19	...	35.49	18.88	12.45	8.22	5.70	37.40	26.00	30.90	5.48	8,060	.2	.3	0	.2	.5	4.0	4.9
89	3	6	18.26	.096	52.97	32.27	22.94	14.98	4.25	32.55	42.52	20.68	3.05	10,147	.2	4.2	0	.2	.5	1.6	4.9
37	2	5	8.50	.704	66.44	42.94	32.11	24.03	6.60	33.80	37.82	21.78	5.06	9,587	.3	.5	0	.2	.3	3.1	4.9
28	2	5	14.12	.282	56.54	27.37	17.30	8.50	6.58	34.07	25.25	34.10	5.09	7,735	0	1.7	0	.2	.3	2.3	4.8
89	2	6	16.70	.095	40.04	22.44	16.95	10.26	5.55	32.30	40.85	21.30	3.80	9,568	.3	4.5	0	.2	.7	2.6	4.7
92	1	6	18.90	...	53.33	26.06	15.10	8.86	6.25	30.87	47.37	15.53	1.43	11,027	.2	.3	0	0	.3	2.8	4.7
62	3	6	21.49	.376	58.81	26.65	20.60	14.53	4.14	34.13	48.54	13.19	1.19	11,603	...	...	0	.3	.7	3.0	4.5
75	3	6	12.31	.830	83.81	69.75	56.71	40.73	6.18	32.42	41.12	20.28	5.79	9,385	.5	3.0	0	.2	.7	3.1	4.5
87	1	6	8.38	.112	43.01	17.57	9.57	4.91	5.43	37.32	32.60	24.65	5.11	9,563	0	2.8	0	0	.2	1.2	4.5

TABLE 2.—*Results of tests of rib dusts*—Continued.

Mine No.	Place of sampling.	Seam.	Results of sizing tests.														...re per square inch of—				
			Air-drying loss.	Dust per square foot.	Dust through a 20-mesh sieve.					V n							Ground dust through a 200-mesh sieve at a coil temperature of—				
																	800° C.	900° C.	1,000° C.	1,100° C.	1,200° C.
1	2	3	4	5	6												18	19	20	21	22
			P. ct.	Ounces.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	B. t. u.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
59	1	6	19.08	.200	86.43	72.86	62.45	45.53	4.75	31.76	47.20	16.29	2.57	11,097	...	...	0	.3	.5	1.8	4.5
9	2	2	10.03	...	38.36	29.68	25.80	21.94	...	...	...	...	...	...	0	.2	0	0	.2	2.3	4.2
46	3	5	12.81	.431	69.64	46.02	30.64	21.81	4.40	28.97	39.18	27.45	3.88	9,543	.5	1.0	0	.2	1.0	1.7	3.9
56	1	6	12.10	.019	48.97	30.46	23.14	15.29	4.09	32.53	46.69	15.79	1.67	11,176	...	...	0	.3	.7	3.1	3.7
38	3	5	4.81	.246	86.60	81.29	77.32	71.60	4.28	28.56	26.44	42.73	3.15	7,022	3.8	5.1	0	.2	.3	1.8	3.6
34	3	5	8.62	.076	34.18	22.95	17.14	13.76	4.88	32.22	25.35	37.55	5.31	7,333	1.6	2.8	0	0	0	3.0	3.6
50	1	6	6.42	.058	67.02	50.28	42.39	32.82	3.65	27.56	38.95	29.85	1.50	9,061	.7	.3	0	.3	.9	1.6	3.3
60	1	6	19.26	.033	60.22	38.54	30.02	22.18	4.21	31.10	52.37	12.32	2.84	11,718	...	...	0	.3	.7	1.6	3.3
41	2	5	3.91	.045	31.00	14.40	9.03	5.61	...	...	...	...	...	...	0	.7	0	.2	.3	1.7	3.3
92	2	6	20.68	...	82.91	57.30	34.69	10.82	5.65	30.75	47.80	15.80	1.68	11,068	.3	.5	0	.2	.5	2.1	3.3
19	2	1	23.30	...	40.64	9.66	5.83	3.35	5.28	35.50	14.82	44.40	6.80	6,325	0	1.9	0	.3	.7	1.4	3.1
57	3	6	8.99	...	66.97	51.69	44.15	35.10	...	...	...	...	...	...	...	...	0	.2	.3	2.1	3.0
65	2	6	20.16	.258	80.80	48.19	36.55	26.66	8.42	34.16	42.71	16.71	4.14	9,997	...	...	0	.3	.5	1.6	3.0
[illegible]	3	6	5.96	.110	17.16	8.94	6.19	4.90	6.55	30.95	31.75	30.75	5.75	7,519	.2	0	0	.2	1.0	2.3	3.0
81	2	6	15.20	.254	59.60	42.30	33.70	21.30	4.35	36.80	33.35	26.50	6.12	8,885	2	2	0	.2	.5	2.4	3.0
75	2	6	8.11	1.067	79.72	58.02	41.98	28.69	8.00	33.00	36.40	22.60	6.08	8,075	.2	.2	0	.2	.7	1.4	2.8
14	2	2	18.81	.009	35.00	10.88	7.08	3.96	4.55	31.30	40.20	23.96	3.54	8,710	0	.7	0	0	.2	.3	2.7
64	1	6	29.10	.462	94.59	68.00	42.66	22.40	3.13	30.66	53.27	12.96	2.30	12,004	...	...	0	.2	.5	1.2	2.6
88	1	6	8.01	.057	69.02	35.45	23.59	14.45	5.18	37.09	31.73	26.00	5.07	8,381	.2	1.4	0	0	0	.2	2.3
10	3	2	6.96	...	65.92	49.74	43.10	39.42	2.90	24.85	21.92	50.33	4.04	6,591	0	.2	0	0	.2	.9	2.3
74	3	6	9.74	.172	20.88	7.39	4.53	3.11	4.20	35.00	28.75	32.05	4.36	7,673	0	2.1	0	0	.2	.7	2.1
57	1	6	14.19	...	67.41	49.09	39.89	29.77	4.70	28.69	44.47	22.14	1.61	10,206	...	...	0	.2	.3	1.4	1.9
52	3	6	4.76	.110	57.00	29.32	22.82	17.48	4.27	32.69	34.38	28.68	5.05	8,827	0	0	0	.3	.4	.6	1.9
93	3	6	11.11	...	83.66	66.62	43.25	19.41	6.07	29.23	34.85	29.85	3.04	8,741	.3	.5	0	0	.2	.7	1.9
24	2	1	9.68	...	24.77	11.60	7.28	3.79	5.00	32.13	28.17	34.70	6.24	7,290	.3	1.4	0	.2	.5	1.4	1.9
73	3	6	8.45	.207	48.57	27.57	19.91	12.26	6.60	35.05	29.37	29.08	7.13	7,692	0	.7	0	0	.2	.7	1.7
38	3	5	3.76	.101	45.45	17.95	12.58	7.34	8.00	32.63	29.94	29.34	5.30	8,077	.3	.7	0	.2	.3	.7	1.6
54	3	6	11.18	.067	78.58	55.27	47.94	36.94	5.45	28.25	35.55	30.75	2.51	8,474	...	...	0	.2	.8	1.1	1.5
58	3	6	7.26	.088	58.17	43.36	39.53	33.99	4.90	31.42	42.31	21.37	1.84	10,229	...	...	0	.2	.3	.7	1.3
85	3	6	5.58	.205	50.36	27.40	22.04	17.14	7.10	33.98	29.30	29.06	5.30	8,267	.3	.5	0	0	.2	.6	1.2

63......	2	6	16.96	.222	78.28	44.43	28.86	17.21	6.32	32.40	31.76	29.52	3.83	8,508	......	......	0	.2	.3	.7	1.2
18......	2	1	10.20	......	73.23	53.02	45.85	29.32	3.90	28.15	17.60	50.35	7.04	5,396	.2	.9	0	.2	.3	.9	1.2
85......	2	6	3.76	.249	80.17	62.96	54.71	44.23	8.98	31.75	34.80	24.47	6.43	8,262	.5	1.2	0	0	.2	.3	1.1
54......	1	6	7.98	.022	86.80	73.11	65.78	50.30	5.06	29.54	46.10	19.30	2.00	10,395	......	......	.2	.2	.3	.9	1.1
86......	3	6	8.15	.312	43.10	19.62	12.71	8.39	5.65	31.85	39.12	23.38	5.88	9,176	.2	.3	0	0	0	.2	1.1
55......	2	6	19.04	.120	54.07	33.82	23.65	13.73	3.23	28.73	44.33	23.71	0.96	10,098	......	......	0	0	.2	.2	1.0
69......	2	6	27.19	.121	75.07	46.19	33.91	21.86	5.75	32.33	37.84	24.08	5.84	8,609	1.0	3.2	0	.2	.2	.4	1.0
66......	2	6	10.48	.306	45.20	21.40	15.97	10.48	5.50	31.76	33.16	29.58	5.97	7,315	0	.2	0	0	.2	.7	1.0
72......	3	6	9.60	......	40.43	21.99	15.34	10.05	6.80	35.38	30.02	26.90	6.15	7,619	0	.2	0	0	.2	.3	1.0
80......	3	6	8.66	.236	41.15	17.11	11.02	5.92	7.13	38.97	26.30	27.60	9.12	7,195	0	.2	0	0	.2	.3	1.0
85......	1	6	5.56	.194	91.30	73.79	57.68	40.35	7.85	32.40	38.32	21.43	6.08	9,148	.3	1.4	0	0	.2	.5	1.0
66......	3	6	......	......	......	(a)	......	......	5.50	31.76	33.16	29.58	5.97	7,315	0	.2	0	0	.2	.7	1.0
27......	3	5	11.06	.201	56.19	42.81	36.31	27.63	5.63	30.87	15.05	48.45	5.05	5,531	.2	.3	0	0	0	0	.9
83......	3	6	9.68	.524	75.51	51.55	39.98	24.91	6.00	32.75	27.82	33.43	5.30	7,144	.2	.7	0	0	0	.3	.9
58......	2	6	5.75	.056	54.76	31.44	26.73	21.79	5.11	36.23	40.46	18.20	1.87	10,278	......	......	0	.2	.3	.7	.9
53......	2	6	14.19	.089	67.99	33.35	22.91	16.44	5.30	31.28	50.91	12.51	.48	11,324	......	......	0	.2	.3	.7	.9
53......	3	6	5.16	.510	54.48	41.08	36.48	33.21	4.72	26.42	47.77	21.09	.56	10,433	......	......	0	.2	.3	.5	.9
56......	2	6	7.16	.125	50.62	29.30	20.93	14.55	5.32	35.35	40.20	19.13	1.52	9,970	......	......	2	.2	.3	.5	.9
59......	2	6	17.67	.070	82.84	67.48	60.31	50.52	5.35	29.36	44.79	20.50	3.05	10,229	......	......	0	.2	.3	.5	.9
62......	1	6	6.77	.302	90.55	85.15	82.55	76.72	4.42	26.15	32.05	37.38	2.74	7,864	......	......	0	.2	.3	.5	.9
65......	3	6	24.68	.208	76.79	53.87	46.32	35.38	5.76	29.99	45.59	18.66	3.38	9,533	......	......	0	.1	.2	.3	.9
68......	2	6	7.52	.185	34.85	21.22	15.96	11.62	4.83	31.82	33.02	30.33	6.88	7,664	0	0	0	0	.2	.3	.9
36......	3	5	8.94	.055	78.09	56.36	41.23	24.83	7.55	33.63	30.84	27.98	5.78	7,992	.3	1.6	0	0	.3	.3	.9
45......	1	5	1.50	.255	61.61	40.80	32.21	22.91	3.90	28.40	43.50	24.20	5.08	9,437	.9	.7	0	0	0	.3	.9
56......	3	6	14.19	.062	52.26	32.94	24.98	18.71	5.45	31.10	42.63	20.82	2.06	10,105	......	......	0	.2	.3	.8	.8
60......	2	6	19.93	.094	87.95	73.32	62.91	49.16	5.55	30.91	46.55	16.99	3.09	10,166	......	......	0	0	.2	.7	.8
64......	2	6	22.02	.464	65.88	27.80	21.85	15.08	5.41	32.76	44.96	16.87	3.14	9,817	......	......	0	0	.2	.5	.8
40......	2	5	13.38	.456	87.28	71.49	53.23	35.02	7.75	31.95	39.30	21.00	5.10	8,545	.3	1.2	0	0	.2	.5	.8
51......	3	6	11.78	.222	71.77	61.23	58.92	56.26	4.37	30.17	40.11	25.35	3.05	9,430	......	......	0	.2	.5	.6	.7
53......	1	6	6.27	.296	81.68	68.83	60.10	47.47	6.48	27.57	49.15	16.80	1.38	10,526	......	......	0	.2	.2	.4	.7
55......	1	6	12.32	.200	81.34	50.77	37.62	24.59	4.78	32.24	44.58	18.40	2.81	9,967	......	......	0	.2	.3	.5	.7
55......	3	6	13.13	.121	87.90	64.25	50.59	32.49	4.38	29.79	43.87	21.96	1.70	9,909	......	......	0	0	.2	.3	.7
58......	1	6	9.61	.052	45.46	27.92	23.63	18.26	4.05	30.34	41.15	24.46	.83	9,857	......	......	0	0	.2	.4	.7
63......	3	6	6.53	.265	88.05	71.67	59.55	41.18	6.20	38.11	21.06	34.63	6.55	7,308	......	......	0	.2	.3	.3	.7
77......	3	6	9.79	.300	52.52	37.21	31.24	23.65	6.35	29.10	29.87	34.68	5.22	7,456	.2	0	0	0	0	.2	.7
84......	1	6	......	......	......	......	......	......	7.43	31.62	23.87	37.08	5.66	6,777	.2	.2	0	0	0	0	.7
87......	2	6	4.34	.233	64.89	38.01	25.93	16.65	8.23	35.87	32.65	23.25	6.82	8,354	.2	1.2	0	0	0	.2	.7
50......	2	6	12.22	.005	47.08	22.06	14.90	7.87	5.15	32.10	37.50	25.25	5.67	9,144	0	.7	0	0	.2	.3	.7
93......	2	6	6.89	......	80.33	59.40	44.51	32.33	6.33	29.67	28.07	35.93	2.93	7,571	.2	.2	0	0	0	.3	.7
51......	1	6	16.00	.225	51.62	22.67	17.32	11.85	4.31	28.61	45.11	21.97	2.37	10,300	......	......	0	.3	.5	.6	.6
51......	2	6	13.59	.408	58.94	31.62	25.19	19.00	4.03	38.56	25.15	32.26	3.90	8,075	......	......	0	.1	.2	.4	.6
60......	3	6	8.91	.228	66.38	49.32	43.50	38.21	5.39	33.42	36.93	24.26	3.63	8,469	......	......	0	.2	.3	.4	.5
67......	2	6	6.82	.444	67.47	47.57	39.19	26.38	7.33	34.38	22.71	35.58	6.24	6,754	.5	.5	0	0	0	0	.5
67......	3	6	5.27	.663	62.76	32.35	24.24	15.63	5.10	25.25	20.85	48.80	3.65	5,483	.2	.7	0	0	0	0	.5
7......	1	2	17.14	......	24.59	10.16	7.06	5.32	1.55	23.38	26.54	48.53	2.49	6,664	0	.2	0	0	0	0	.5
37......	3	5	10.57	.710	68.86	45.13	34.65	25.41	6.63	32.17	31.67	29.53	5.59	7,355	.2	.2	0	0	0	0	.5
44......	3	5	8.21	.139	56.15	35.21	26.24	14.70	3.15	27.03	32.54	37.28	5.20	6,968	.2	.2	0	0	0	.3	.5
91......	3	6	12.00	......	76.90	55.63	39.63	21.72	6.28	27.12	29.65	36.95	2.93	7,418	.3	.3	0	0	0	.3	.5
82......	2	6	9.94	.217	52.08	33.27	27.21	14.75	4.75	37.93	18.14	39.18	7.54	5,879	.2	.5	0	0	0	0	.4
49......	2	5	7.48	.100	56.25	32.33	22.40	15.26	6.20	34.15	29.15	30.50	9.53	6,961	0	.5	0	0	0	0	.3
49......	3	5	8.40	.246	69.43	57.59	52.41	43.54	3.98	25.77	26.60	43.65	7.21	5,760	.2	.2	0	0	0	0	.3

a No data.

TABLE 2.—*Results of tests of rib dusts*—Continued.

Mine No.	Place of sampling.	Seam.	Air-drying loss.	Dust per square foot.	Results of sizing tests. Dust through a 20-mesh sieve.	Dust through a 60-mesh sieve.	Dust through a 100-mesh sieve.	Dust through a 200-mesh sieve.	Proximate analysis. Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Calorific value.	Pressure per square inch of— Dust through a 20-mesh sieve at a coil temperature of 1,200° C.	Dust through a 200-mesh sieve at a coil temperature of 1,200° C.	Ground dust through a 200-mesh sieve at a coil temperature of— 800° C.	900° C.	1,000° C.	1,100° C.	1,200° C.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
			P. ct.	*Ounces.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*
69	3	6	9.24	.469	60.92	38.15	31.68	26.62	5.23	24.22	22.22	48.33	5.55	4,511	.2	.2	0	0	0	0	0.3
70	3	6	4.79	...	...	...	...	...	10.53	31.57	22.37	35.53	6.75	5,179	0	.2	0	0	0	0	.3
74	2	6	8.32	.425	18.56	6.64	4.18	2.55	5.50	36.30	18.20	40.00	5.35	6,309	0	.2	0	0	0	0	.3
78	3	6	9.24	.283	39.13	17.02	10.33	5.20	5.13	37.07	26.65	31.15	6.83	6,714	0	0	0	0	0	0	.3
81	3	6	5.05	.168	39.95	27.44	22.00	18.09	5.23	30.27	30.25	34.25	6.24	7,160	.2	.2	0	0	0	0	.3
82	3	6	11.63	.466	59.94	35.88	24.95	12.73	6.25	39.28	14.92	39.55	9.02	4.990	0	0	0	0	0	0	.3
83	1	6	9.30	.293	63.85	38.18	28.15	17.67	6.68	34.27	16.75	42.30	6.71	5,182	0	.2	0	0	0	0	.3
15	3	2	12.55	...	51.84	26.94	19.24	10.39	4.60	34.13	31.99	29.28	5.75	7,114	.3	.5	0	0	0	0	.3
32	3	5	9.03	.785	73.78	44.86	30.42	18.15	9.23	39.17	18.52	33.08	8.14	6,046	0	3.7	0	0	0	0	.3
40	3	5	7.49	.137	69.20	53.59	47.64	42.38	5.78	27.55	21.39	45.28	5.51	5,540	.2	.2	0	0	0	0	.3
42	3	5	8.31	.097	57.99	46.18	43.09	13.38	...	29.35	...	...	6.70	5,497	0	.2	0	0	0	0	.3
43	2	5	19.69	.193	63.49	22.74	13.23	8.14	7.25	37.13	17.82	37.80	9.74	5,661	.2	.7	0	0	0	0	.3
45	2	5	8.81	.217	61.89	20.82	18.20	10.82	5.28	35.09	29.45	30.18	9.20	7,000	0	.2	0	0	0	0	.3
46	2	5	2.39	.268	43.09	29.39	21.25	13.29	4.18	28.37	32.17	35.28	6.61	7,524	.5	.5	0	0	0	0	.3
100	3	3	4.10	...	16.13	8.09	4.22	2.98	...	...	...	...	...	...	.2	.5	0	0	0	0	.3
79	3	6	12.10	.554	48.30	26.10	19.00	12.20	5.90	33.20	25.47	35.43	6.60	6,181	0	.9	0	0	0	0	.3
91	2	6	9.37	...	74.44	55.62	41.82	28.58	5.65	26.05	20.30	48.00	2.96	6,008	.3	.3	0	0	0	0	.3
92	3	6	6.06	...	62.38	46.12	34.41	24.00	5.58	25.92	19.95	48.55	2.94	5,555	.2	.2	0	0	0	0	.2
47	2	5	9.07	.109	50.91	16.06	10.20	5.78	4.30	43.22	22.68	29.80	7.89	5,931	0	0	0	0	0	0	.2
59	3	6	7.15	.264	86.13	70.91	64.93	59.60	4.32	26.31	28.04	41.33	2.41	6,851	...	...	0	0	0	0	.2
89	1	6	16.58	.236	51.86	32.05	20.41	9.71	6.50	37.55	11.27	44.68	6.79	4,333	0	0	0	0	0	0	.2
11	3	2	4.23	...	66.68	64.93	64.48	64.16	2.25	21.32	19.50	56.93	2.69	4,666	0	0	0	0	0	0	.2
13	3	2	12.67	.140	64.45	43.35	31.48	21.61	5.23	31.37	18.27	45.13	6.28	4,444	.2	0	0	0	0	0	.2
39	3	5	3.95	.633	44.84	32.05	23.75	18.38	8.15	31.80	15.95	44.10	6.97	4,711	1.0	.2	0	0	0	0	.2
41	3	5	6.24	.169	36.43	21.48	16.27	12.30	7.10	30.78	8.99	53.13	8.72	3,724	0	0	0	0	0	0	.2
42	1	5	4.66	.024	26.15	13.45	8.51	6.11	...	...	...	...	...	...	0	0	0	0	0	0	.2
48	3	5	3.61	.675	47.07	31.37	17.25	12.48	5.85	30.50	26.12	37.53	5.45	5,788	0	.3	0	0	0	0	.2
97	3	7	48.02	...	...	18.81	8.50	4.39	...	...	...	...	...	...	...	...	0	0	0	0	0
1	2	2	3.49	.337	27.00	18.56	15.91	12.67	2.63	17.30	6.47	73.60	3.85	2,455	0	0	0	0	0	0	0
11	2	2	2.09	...	22.82	15.47	13.83	13.03	1.30	11.85	6.35	80.50	1.01	1,757	0	0	0	0	0	0	0

1.......	8	2	2.95	.330	18.54	10.70	9.19	8.31	2.75	16.30	2.55	78.40	4.13	1,022	0	0	0	0	0	0	0
10.......	2	2	3.12	....	28.18	14.10	9.44	6.50	1.85	10.90	3.30	83.05	3.06	1,208	0	0	0	0	0	0	0
2.......	1	2	7.51	....	36.70	18.22	17.42	14.73	7.80	28.15	4.47	59.58	10.42	1,438	0	0	0	0	0	0	0
42.......	2	5	3.56	.088	8.40	3.28	2.20	1.51	....	....	....	....	....	....	0	.2	0	0	0	0	0
4.......	2	2	2.78	....	53.18	45.47	42.82	41.05	1.28	11.40	2.14	85.18	.95	997	0	0	0	0	0	0	0
4.......	3	2	2.94	....	18.58	10.16	8.08	7.10	1.75	10.75	1.47	86.03	1.07	880	0	0	0	0	0	0	0
5.......	2	3	6.03	.180	14.17	8.83	7.62	6.67	....	....	....	....	....	....	0	0	0	0	0	0	0
5.......	8	3	3.92	.252	30.06	19.01	14.94	12.06	1.98	17.87	10.50	69.65	1.90	3,837	0	0	0	0	0	0	0
7.......	2	2	5.73	....	25.15	15.01	13.02	11.71	.65	8.68	1.12	89.55	.29	585	0	0	0	0	0	0	0
7.......	8	2	2.37	.221	22.47	13.30	10.37	8.57	2.53	14.02	0	83.45	2.56	1,003	0	0	0	0	0	0	0
8.......	2	2	2.13	....	89.36	27.71	28.93	21.82	2.10	13.60	5.52	78.78	3.76	1,454	0	0	0	0	0	0	0
8.......	8	2	2.74	....	27.86	20.57	18.09	17.03	2.20	15.05	6.60	76.15	3.01	1,701	0	0	0	0	0	0	0
9.......	3	2	3.02	....	37.36	29.39	27.07	25.21	1.60	14.40	8.60	75.40	2.59	2,237	0	0	0	0	0	0	0
100.....	2	3	2.90	.065	22.01	13.64	10.76	5.83	5.90	21.85	2.17	70.08	6.70	1,845	0	0	0	0	0	0	0

COMMENTS ON TABULATED DATA.

Compared with the fine dusts the 20-mesh samples developed relatively low pressures. The highest pressure recorded on this class of samples was 3.8 pounds. Eight samples out of 190 gave pressures of 1 pound or more. Approximately 40 per cent of the 20-mesh samples developed no measurable pressure, and 80 per cent of the samples gave pressures of less than 0.5 pound. In the small laboratory apparatus the low relative inflammability of the 20-mesh samples is, of course, due to the large proportion of coarse particles that do not take part in the explosion.

As regards most of the samples the inflammability of the 200-mesh part of the original sample did not differ greatly from the inflammability of the part of the original sample ground to pass a 200-mesh sieve. About 70 per cent of the 200-mesh parts gave pressures of 0.5 pound or more, and might therefore give rise to explosions.

As regards the relation between the source of the sample in the mine and the inflammability of the dust, the greater the distance from the working face the higher is the ash content and the lower the inflammability of the sample. The room samples are the most inflammable, and the dust from the main entries is the least inflammable. The 200-mesh part of the room sample from each of 59 out of 70 mines developed a pressure of 0.5 pound, and of these every one but two developed a pressure of 1 pound or more. Rib-dust samples collected near the working face had practically the same composition and inflammability as the face samples.

QUANTITY OF RIB DUST IN MINES.

It is important to know whether enough dust adheres to the ribs to propagate an explosion by itself, that is, without the assistance of gas or of road dusts.

Taffanel has found that the minimum dust density that will regularly propagate an explosion is 0.07 ounce per cubic foot (70 grams per cubic meter), though in one test he obtained propagation with a density as low as 0.023 ounce per cubic foot. In experiments in the 100-foot gallery at the Bureau of Mines experiment station at Pittsburgh, Pa., propagation occurred in two instances with a dust density of 0.032 ounce per cubic foot. If it be assumed that all the dust on the rib is put into suspension, and if the average height and width of a room be taken as 7 feet and 24 feet, then in no case was the quantity of dust 20-mesh fine found on ribs in rooms sufficient to produce a density of 0.07 ounce per cubic foot. However, large quantities of fine coal dust resting on the floor or on the gob in rooms might be dangerous, even though the quantity of rib dust was small.

As regards the dust fine enough to pass a 20-mesh screen, for about one-fourth of the samples of rib dust collected in the entries, the quantity was sufficient to yield a dust density of 0.07 ounce per cubic foot, if the entries are assumed to be 7 feet high and 8 feet wide.

In drawing conclusions from the results obtained in the study of the rib dusts, the reader should remember that the deposits of dust on the ribs vary greatly in character and in quantity, from point to point in a mine, and that the results obtained from two or three samples are not necessarily representative of the entire mine.

INFLAMMABILITY OF MINE-ROAD DUSTS.

TABULATED RESULTS.

The results of the laboratory examination of the road dusts are presented in Table 3. The data are arranged in the same manner as the rib-dust data in Table 2.

TABLE 3.—*Results of analyses and of inflammability tests of road dusts.*

Mine No.	Place of sampling.[a]	Seam.	Air-drying loss.	Results of sizing tests.				Approximate analysis.					Calorific value.	Pressure from—						
				Dust through a 20-mesh sieve.	Dust through a 60-mesh sieve.	Dust through a 100-mesh sieve.	Dust through a 200-mesh sieve.	H_2O.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.		Dust through a 20-mesh sieve at a coil temperature of 1200 °C.	Dust through a 200-mesh sieve at a coil temperature of 1200° C.	Ground dust through a 200-mesh sieve at a coil temperature of— 800° C.	900° C.	1000° C.	1100° C.	1200° C.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
			P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*
71	2	6	6.01	61.94	39.20	28.34	21.60	3.53	35.17	36.97	24.33	4.00	9,763	...	...	0	0.7	1.0	7.2	7.9
43	1	5	6.32	35.12	19.18	13.33	8.13	3.40	30.35	46.56	19.69	2.80	10,431	0	1.4	0	.2	1.2	5.8	6.8
66	2	6	13.07	25.87	11.01	6.99	3.67	3.13	33.92	39.63	23.32	3.93	10,049	0	3.9	0	.3	2.4	5.9	6.8
88	1	6	2.93	38.04	20.74	12.91	9.89	4.73	35.42	38.60	21.25	3.12	10,359	...	...	0	.3	1.6	6.0	6.8
49	1	5	4.22	49.43	28.96	22.73	16.06	2.53	31.87	41.40	24.20	4.90	10,089	.4	2.3	0	.3	1.2	5.8	6.6
75	2	6	9.10	39.20	22.50	17.20	11.60	3.35	28.10	29.73	38.82	3.49	7,731	.2	2.3	0	.3	1.7	5.2	6.5
85	2	6	6.73	42.23	23.02	16.89	12.26	5.58	30.37	37.37	26.68	3.30	9,117	...	...	0	.2	1.2	5.8	6.5
63	1	6	2.81	39.39	22.06	17.00	12.48	5.33	34.08	43.66	16.93	2.67	10,895	...	...	0	.5	.9	3.7	6.3
76	2	6	7.38	39.28	20.60	14.56	10.44	4.05	30.35	33.90	31.70	3.52	8,674	0	1.4	0	.3	1.6	5.4	6.3
14	1	2	13.52	15.63	2.47	1.23	.61	3.13	29.07	44.62	23.18	1.81	10,363	0	1.4	0	.3	1.6	5.6	6.2
80	2	6	2.40	33.94	15.86	12.30	8.61	5.05	32.48	34.22	28.25	3.24	9,081	...	...	0	.2	1.0	5.2	6.2
6	2	2	...	...	...	...	...	2.75	29.73	25.07	42.45	2.06	7,484	...	...	0	.2	1.1	5.1	5.9
84	1	6	6.54	62.34	42.37	33.54	25.53	4.20	30.35	32.78	32.67	3.74	8,314	...	...	0	.5	1.7	4.9	5.9
13	1	2	11.17	14.25	4.82	3.14	1.88	3.68	30.12	49.57	16.63	1.53	11,232	.0	1.4	0	.3	1.2	4.9	5.8
15	2	2	9.72	1.26	.35	.18	.10	2.30	27.60	42.02	28.08	2.35	9,725	.2	1.4	0	.2	1.2	4.5	5.8
86	2	6	5.27	43.10	21.92	15.61	10.48	4.25	28.30	32.20	35.25	3.35	8,231	.2	<.9	0	.2	1.0	5.1	5.8
32	1	5	9.13	44.59	21.90	14.21	9.26	5.35	31.10	35.10	28.45	4.44	8,957	.4	3.9	0	.2	1.2	4.5	5.6
66	1	6	8.53	37.91	23.48	18.27	14.14	4.05	32.50	35.40	28.05	4.58	8,467	.2	<.9	0	0	.7	3.8	5.6
68	2	6	...	...	...	...	...	5.08	30.52	32.90	31.50	3.80	8,240	...	...	0	.2	1.2	4.9	5.6
38	2	5	5.20	37.47	24.22	18.11	10.66	5.30	31.40	30.05	33.25	3.59	8,307	0	4.4	0	.2	1.2	4.0	5.4
43	2	5	2.93	46.63	21.18	14.99	11.27	1.93	24.82	35.80	37.45	2.64	8,446	0	1.4	0	0	.5	4.0	5.4
87	2	6	4.19	41.25	22.29	17.08	11.97	4.23	32.12	36.96	26.69	3.50	9,497	0	1.4	0	.2	.7	4.4	5.4
3	1	2	6.98	51.46	31.13	23.26	15.93	3.65	27.85	26.10	42.40	3.95	6,979	0	<.9	0	0	.9	4.4	5.2
26	2	5	9.40	32.18	18.34	13.52	9.69	3.20	30.55	26.40	39.85	3.18	7,763	0	1.4	0	.2	1.0	3.3	5.2
65	1	6	10.73	24.18	9.61	6.93	4.81	4.60	32.14	39.70	23.56	2.45	10,174	.5	4.4	0	.2	.7	3.0	5.2
77	1	6	6.01	32.56	19.42	14.88	11.16	4.63	28.62	25.88	40.87	4.01	6,973	0	<.9	0	0	1.0	3.8	5.2

75......	1	6	8.00	51.10	36.00	18.50	11.60	4.50	27.35	32.35	35.80	4.69	7,420	0	< .9	0	.2	.7	3.7	5.0
11......	2	2	7.77	48.19	20.11	23.31	17.16	2.18	34.77	17.65	45.40	3.20	6,936	......	......	0	.2	.5	3.3	4.7
77......	2	6	7.90	44.99	23.93	18.33	13.00	4.23	27.57	29.56	38.64	3.50	7,472	0	< .9	0	0	.7	3.5	4.7
88......	2	6	4.01	30.99	16.07	12.68	7.97	4.55	29.73	35.82	29.90	2.93	8,937	......	......	0	.2	1.2	3.3	4.7
29......	2	5	14.24	63.09	38.19	30.63	7.56	3.00	28.60	24.30	44.10	2.99	7,205	0	1.4	0	0	.5	3.2	4.5
57......	2	6	4.70	41.61	20.06	14.82	10.84	3.95	26.27	40.61	29.17	1.02	9,331	......	......	0	0	.5	2.8	4.5
73......	2	6	5.95	48.91	25.95	18.63	13.64	4.33	26.72	31.05	37.90	3.42	7,394	0	< .9	0	0	.7	3.0	4.5
85......	1	6	5.29	41.56	24.28	18.31	12.73	6.66	30.87	39.82	22.65	3.73	9,364	0	< .9	0	0	.5	3.0	4.5
90......	1	6	6.68	28.09	12.45	8.03	4.16	3.82	27.16	31.83	37.19	3.49	7,810	......	......	0	0	.7	3.5	4.5
37......	1	5	9.67	45.16	25.07	18.28	12.27	4.75	28.85	30.90	35.50	4.24	7,553	.2	< .9	0	0	.7	3.4	4.4
83......	1	6	3.36	47.97	28.58	21.89	15.34	5.85	29.80	31.27	33.08	3.42	7,970	0	1.4	0	0	.5	2.8	4.4
65......	2	6	13.53	21.56	11.12	9.21	7.30	4.42	32.39	41.40	21.79	2.88	10,289	.2	< .9	0	.2	1.2	2.8	4.2
45......	2	5	6.42	46.03	24.73	17.69	12.45	3.10	28.05	45.62	23.23	3.07	10,327	0	< .9	0	0	.5	3.3	4.0
80......	1	6	3.22	37.68	20.50	15.44	12.25	5.72	29.58	28.25	36.45	4.04	7,621	......	......	0	0	.5	3.1	4.0
72......	2	6	9.24	51.85	30.25	21.29	14.81	5.30	29.35	34.10	31.26	3.80	8,024	0	< .9	0	.2	.5	2.3	4.0
74......	2	6	4.99	37.70	22.67	16.94	12.76	7.18	31.07	33.45	28.30	4.79	7,792	0	< .9	0	0	.3	1.4	3.8
9......	2	2	7.95	49.13	29.17	22.84	17.08	2.65	26.65	23.00	47.70	4.23	6,167	0	< .9	0	0	.3	1.4	3.7
27......	2	5	12.06	56.93	31.66	22.96	15.62	2.88	26.87	19.05	51.20	2.83	5,931	0	1.4	0	.2	.3	1.6	3.7
28......	2	5	11.22	73.10	50.00	23.10	15.34	2.83	29.77	19.22	48.18	2.81	6,244	0	< .9	0	0	.3	2.1	3.3
67......	1	6	4.71	39.12	23.56	17.36	13.12	6.00	28.90	31.25	33.85	3.79	7,45[illegible]	0	< .9	0	0	.5	1.4	3.3
57......	1	6	11.19	23.53	8.07	5.21	3.19	4.73	28.65	44.37	22.25	.98	10,085	......	......	0	0	.3	1.1	3.1
37......	2	5	7.94	32.07	13.62	8.10	4.48	4.38	24.72	25.67	45.23	3.51	6,680	0	1.4	0	0	.2	1.0	3.0
74......	1	6	6.34	39.41	21.86	15.63	12.15	6.68	30.02	34.35	28.95	4.40	8,096	0	< .9	0	0	.2	1.2	3.0
26......	2	5	9.69	44.21	25.54	18.56	12.61	2.38	23.37	17.05	57.20	2.74	5,396	0	1.4	0	0	.2	1.0	2.9
34......	2	5	10.91	54.95	29.01	21.08	13.87	4.23	26.62	24.47	44.68	3.94	6,494	0	< .9	0	.2	.3	.6	2.8
28......	1	5	11.83	26.40	10.60	6.27	3.46	3.15	26.40	21.85	48.60	2.79	6,403	0	< .9	0	0	.2	.7	2.5
60......	1	6	6.79	34.00	15.14	10.28	8.43	7.02	31.42	47.53	14.03	1.67	10,931	.2	< .9	0	.2	.9	1.0	2.3
68......	1	6	......	......	......	......	......	7.58	34.22	28.55	29.65	4.01	7,643	......	......	0	0	.3	1.6	2.3
59......	1	6	3.70	43.55	24.62	18.81	13.82	5.83	31.20	44.60	18.37	1.60	10,762	0	< .9	0	.2	.7	.9	2.3
10......	2	2	8.40	38.72	23.72	17.03	12.37	2.95	23.80	19.12	54.13	3.57	5,375	0	< .9	0	0	.2	.7	2.3
6......	1	2	......	......	......	......	......	2.28	22.52	21.77	53.43	1.61	5,773	0	< .9	0	0	.2	.9	2.2
56......	2	6	4.27	45.61	25.03	18.15	12.80	5.18	31.32	40.58	22.92	1.65	10,166	......	......	0	.2	.4	.9	2.1
71......	1	6	6.22	34.58	20.00	15.42	11.08	4.13	29.02	27.45	39.40	6.46	6,809	0	6.2	0	0	.2	1.0	2.1
33......	2	5	9.83	37.91	19.65	13.61	9.39	3.90	27.00	23.90	45.20	3.32	6,743	0	1.4	0	0	.2	1.0	2.1
25......	1	5	10.78	38.64	21.80	16.49	11.52	3.64	24.81	23.34	48.21	4.30	6,311	0	< .9	0	0	.2	.3	2.1
49......	2	5	3.39	32.04	17.17	11.36	7.61	2.33	21.32	24.33	52.02	3.46	6,145	0	< .9	0	0	.2	.4	1.9
59......	2	6	4.28	30.31	25.83	20.43	15.27	5.59	28.15	39.34	26.92	1.39	9,383	0	1.4	0	.3	.5	.8	1.9
5......	2	2	9.77	52.87	37.27	33.33	28.73	1.88	21.92	20.72	55.48	1.46	5,553	......	......	0	0	.0	.3	1.7
39......	2	5	5.41	48.20	26.96	20.36	14.93	5.03	26.72	21.50	46.75	3.80	6,338	0	< .9	0	0	.0	.3	1.7
53......	1	6	4.54	27.09	12.83	9.35	6.49	6.64	29.92	45.73	17.71	.46	10,726	......	......	0	.2	.3	.5	1.7
63......	2	6	5.86	44.95	25.90	21.66	15.63	5.43	29.99	42.39	22.19	1.50	10,134	......	......	0	.3	.7	1.0	1.6
70......	2	6	5.06	46.22	25.13	18.88	12.37	4.38	27.52	25.75	42.35	4.46	6,514	.2	< .9	0	0	.0	.5	1.6
92......	1	6	7.66	22.77	9.25	8.06	2.01	4.48	22.65	29.87	43.00	2.00	6,953	0	.2	0	0	.2	.7	1.6
91......	2	6	6.40	34.63	18.31	5.89	2.84	4.90	24.13	27.87	43.10	2.01	6,943	0	.2	0	0	.2	.7	1.4
48......	1	5	12.64	18.30	7.02	5.10	3.40	2.63	20.17	27.32	49.88	1.87	6,329	0	< .9	0	0	.1	.5	1.2
83......	2	6	4.74	34.60	16.35	11.25	7.06	6.30	26.40	25.20	42.10	3.74	6,430	0	< .9	0	0	0	.3	1.2
89......	2	6	16.25	43.10	30.40	25.70	21.20	2.12	14.43	11.42	72.03	1.08	3,002	......	......	0	0	0	.3	1.2
21......	2	3	4.21	51.18	32.65	25.27	17.58	2.90	22.80	24.50	49.80	2.60	6,194	0	< .9	0	0	0	.3	1.0
29......	1	5	7.13	75.55	52.21	28.37	18.96	3.15	24.25	15.95	56.65	4.16	5,049	0	< .9	0	0	.2	.5	1.0
53......	2	6	4.51	40.97	19.72	14.58	10.27	5.36	28.24	41.97	24.43	.50	9,868	......	......	0	0	.3	.5	1.0

[a] Figure 1 indicates that sample was taken from the secondary haulageway, and figure 2, that sample was taken from the main haulageway.

TABLE 3.—*Results of analyses and of inflammability tests of road dusts*—Continued.

											analysis.			Pressure from—						
											Ash.	Sulphur.	Calorific value.	Dust through a 20-mesh sieve at a coil temperature of 1200° C.	Dust through a 200-mesh sieve at a coil temperature of 1200° C.	Ground dust through a 200-mesh sieve at a coil temperature of—				
																800° C	900° C.	1000° C.	1100° C.	1200° C.
											12	13	14	15	16	17	18	19	20	21
											P. ct.	P. ct.	B. t. u.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.
											49.08	5.21	5,348	0	.3	0	0	.2	.5	1.0
											37.78	1.56	7,693	0	.2	0	0	0	.3	1.0
											58.76	3.95	4,835	0	.2	0	0	0	0	.9
											42.25	3.05	6,647	.2	<.9	0	0	0	.2	.9
											45.37	3.48	6,343	.2	1.4	0	0	0	.2	.9
											49.63	5.11	5,020	0	<.9	0	0	.2	.5	.9
											48.40	1.26	6,235	0	.2	0	0	0	.3	.7
											37.98	3.18	7,628	0	<.9	0	0	0	.2	.7
											34.98	1.65	8,2[illegible]			0	0	.2	.3	.7
											56.05	2.51	5,110	0	<.9	0	0	0	.3	.6
											49.23	3.64	4,374	0	0	0	0	0	0	.5
											41.40	2.96	7,141	0	<.0	0	0	0	0	.5
											52.75	3.49	4,855	0	<.9	0	0	0	0	.5
											51.56	3.36	5,051	0	<.9	0	0	0	0	.5
											63.35	3.03	3,841	0	<.9	0	0	0	0	.5
											48.06	3.69	5,657	0	<.9	0	0	0	0	.5
											48.60	2.50	6,048	0	<.9	0	0	0	0	.6
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	57.00	2.67	4,563	0	<.9	0	0	0	0	.5
44......	1	5	4.91	16.34	5.16	3.07	1.72	2.25	19.40	25.60	52.75	4.78	5,656	0	<.9	0	0	0	0	.5
67......	2	6	6.62	36.93	18.73	14.29	9.84	4.80	24.25	23.02	47.93	3.76	5,409	0	<.9	0	0	0	0	.5
79......	1	6	6.59	42.55	12.55	6.67	3.33	3.46	22.97	15.65	57.90	4.38	4,720	0	2.8	0	0	0	0	.5
87......	1	6	3.83	43.07	24.95	19.26	12.80	5.45	26.00	29.67	38.68	4.40	6,709	0	<.9	0	0	0	0	.5
44......	2	5	5.45	28.37	9.56	5.91	3.18	1.56	16.30	20.60	61.55	1.67	5,116	0	<.9	0	0	0	0	.4
1......	1	2	4.52	42.39	27.64	21.37	16.99	2.63	17.67	11.20	68.50	3.53	3,425			0	0	0	0	.3
2......	1	2	7.50	47.38	27.86	20.94	15.06	2.65	20.56	18.27	58.53	3.01	4,711	0	<.9	0	0	0	0	.3
9......	1	2	7.37	33.29	19.10	13.52	10.21	2.78	20.82	15.80	61.15	3.72	3,902	0	<.9	0	0	0	0	.3

10	1	2	8.47	44.21	23.05	18.53	13.80	1.75	15.00	7.42	75.83	3.93	2,444	0	< .9	0	0	0	0	.3
13	2	2	9.10	38.78	11.35	6.34	1.83	1.77	12.93	14.11	71.19	.86	3,730	.2	< .9	0	0	0	0	.3
21	1	3	42.96	43.51	33.45	21.94	9.86	2.00	12.00	20.00	77.80	1.36	2,430	0	< .9	0	0	0	0	.3
26	1	5	7.45	33.61	20.13	13.29	7.77	3.70	20.45	12.15	63.70	3.77	3,940	0	< .9	0	0	0	0	.3
31	1	5	8.48	35.21	18.53	13.65	10.14	3.70	21.00	18.85	56.45	4.04	5,025	0	< .9	0	0	0	0	.3
38	1	5	7.76	39.43	21.93	16.02	11.14	4.21	19.96	17.84	57.99	3.99	4,183	0	< .9	0	0	0	0	.3
39	1	5	7.42	37.02	19.66	14.22	9.64	6.30	21.25	16.85	55.60	4.42	4,149	0	< .9	0	0	0	0	.3
47	1	5	1.49	1.99	.59	.31	.18	3.33	20.27	18.15	58.25	3.15	4,950	0	< .9	0	0	0	0	.3
56	1	6	3.12	56.20	37.84	30.64	24.19	4.97	23.91	27.07	44.05	1.11	6,811	......	......	0	0	0	.2	.3
60	1	6	7.64	42.23	22.17	16.00	10.38	4.60	20.10	19.57	55.73	3.20	4,671	0	< .9	0	0	0	0	.3
73	1	6	8.43	17.78	7.32	4.60	2.98	5.25	22.80	13.50	58.45	4.21	4,221	0	< .9	0	0	0	0	.3
78	1	6	8.72	27.40	10.53	6.60	3.51	5.03	18.67	16.72	59.58	2.95	4,054	0	< .9	0	0	0	0	.3
79	2	6	......	......	......	......	......	1.85	11.85	10.21	76.09	1.32	3,116	......	......	0	0	0	0	.3
19	1	1	42.87	20.77	4.28	1.81	.90	3.53	22.70	6.09	67.68	3.67	3,616	0	.2	0	0	0	0	.3
92	2	6	13.73	16.63	4.83	1.79	.35	5.00	25.58	30.59	38.83	1.54	7,306	0	.1	0	0	0	0	.3
93	2	6	16.92	11.52	3.70	2.88	1.44	2.73	20.10	15.74	61.43	1.03	4,437	0	0	0	0	0	.2	.3
1	2	2	4.71	49.84	30.89	23.68	17.43	2.23	12.47	4.45	90.85	2.78	1,285	......	......	0	0	0	0	.3
93	1	6	6.62	42.54	27.75	15.03	3.66	3.28	18.02	18.52	60.18	1.50	4,622	0	0	0	0	0	0	.2
97	1	6	5.75	42.52	25.12	9.66	6.00	4.03	18.25	18.94	58.78	1.51	4,759	0	0	0	0	0	0	.2
4	1	2	4.87	29.88	17.66	12.05	9.92	1.33	10.70	2.02	85.95	.76	1,055	......	......	0	0	0	0	.2
4	2	2	5.11	55.44	36.81	29.07	23.23	2.05	19.85	14.77	63.33	2.12	4,111	0	< .9	0	0	0	0	.2
5	1	2	9.78	42.55	31.10	28.01	27.77	1.40	13.65	8.35	76.60	.91	2,488	......	......	0	0	0	0	.2
7	1	2	7.62	77.44	27.61	21.89	6.27	1.85	14.35	8.72	75.08	1.84	2,228	......	......	0	0	0	0	.2
7	2	2	5.71	40.15	24.84	17.57	14.84	1.88	14.72	7.92	75.48	1.67	2,261	......	......	0	0	0	0	.2
8	1	2	2.79	28.59	15.10	11.36	7.86	1.95	13.05	5.70	79.30	3.41	1,712	0	< .9	0	0	0	0	.2
8	2	2	4.82	44.41	26.23	20.12	15.94	2.40	19.30	15.85	62.45	3.34	3,967	0	< .9	0	0	0	0	.2
11	1	3	7.11	47.59	25.84	20.14	16.93	2.23	18.57	16.55	62.65	2.21	4,235	......	......	0	0	0	0	.2
12	1	2	16.06	16.44	5.39	3.50	1.89	2.15	13.80	14.80	69.25	1.32	3,377	0	< .9	0	0	0	0	.2
36	2	5	5.93	42.03	21.92	15.91	10.66	5.50	19.80	16.22	58.48	3.63	3,915	0	< .9	0	0	0	0	.2
72	1	6	8.47	44.21	25.05	18.53	13.89	4.70	21.40	14.40	59.50	3.67	3,913	0	< .9	0	0	0	0	.2
76	1	6	......	......	......	......	......	4.65	12.55	2.55	80.25	3.03	1,303	0	< .9	0	0	0	0	.2
89	1	6	18.65	35.30	21.35	18.30	14.60	2.93	17.57	13.60	65.90	1.27	3,757	......	......	0	0	0	0	.2
100	2	6	5.77	42.86	23.70	17.27	11.77	3.35	20.85	13.85	61.95	4.06	3,548	0	0	0	0	0	0	.2
34	1	5	6.52	36.33	18.89	14.28	9.67	4.83	15.97	5.40	73.80	5.09	1,939	0	< .9	0	0	0	0	0
100	1	6	6.20	38.82	18.75	12.86	8.41	2.85	15.05	2.75	79.35	4.18	1,096	0	< .9	0	0	0	0	0
95	1	6	25.07	10.82	5.11	2.75	1.17	5.15	12.48	3.74	78.63	.68	1,693	0	0	0	0	0	0	0

COMMENTS ON TABULATED DATA.

The average proportion of each road-dust sample that was fine enough to pass a 20-mesh screen was 39 per cent. With few exceptions the minimum proportion passing a 20-mesh screen was 20 per cent. Less than half of the dust through a 20-mesh sieve, an average of 15.5 per cent of the original samples, passed a 100-mesh screen, and 10.5 per cent passed a 200-mesh screen.

The last column of Table 3 gives the pressures obtained at 1,200° C. with parts of the road dusts that were ground to pass a 200-mesh screen. A comparison of these values with those in the last column of Table 2 shows that the road dusts are relatively less inflammable than the rib dusts—a result that one would expect because of the

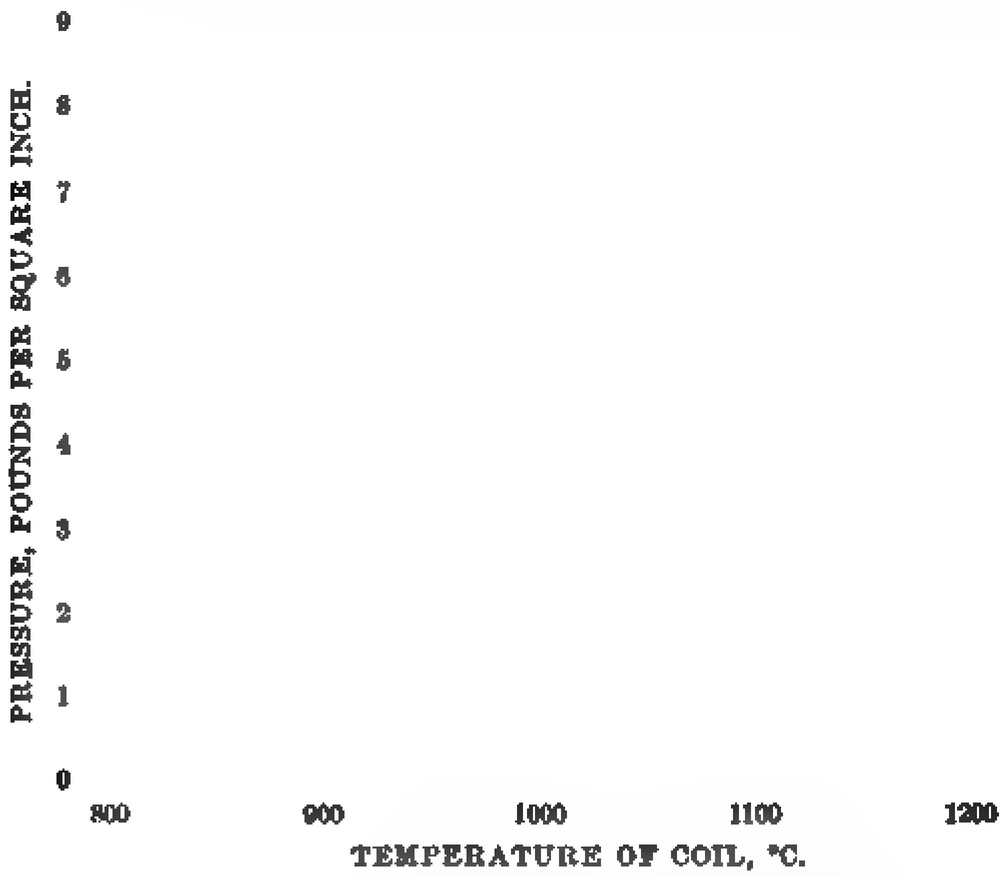

FIGURE 9.—Inflammability curves for all samples from mine No. 8.

higher ash content of the road dusts. The greater number of the ground parts of the road dusts, however, developed pressures of 0.5 pound or more, indicating that the material composing the road dusts of many Illinois mines, when reduced to the proper degree of fineness, can propagate and even give rise to a dust explosion.

The highest pressure developed in the inflammability apparatus by the 20-mesh samples was 0.5 pound, and in the majority of tests no pressure was developed. As the road dusts contained such a small proportion of dust fine enough to pass a 200-mesh screen, it is not surprising that their inflammability is so low.

The pressures developed by the 200-mesh parts of the road-dust samples are reported in column 16. In most of these samples the

pressure is reported as <0.9. Most samples that did not develop sufficient pressure (0.9 pound per square inch) to raise a weight of 25 grams were not tested with smaller weights on the pressure-

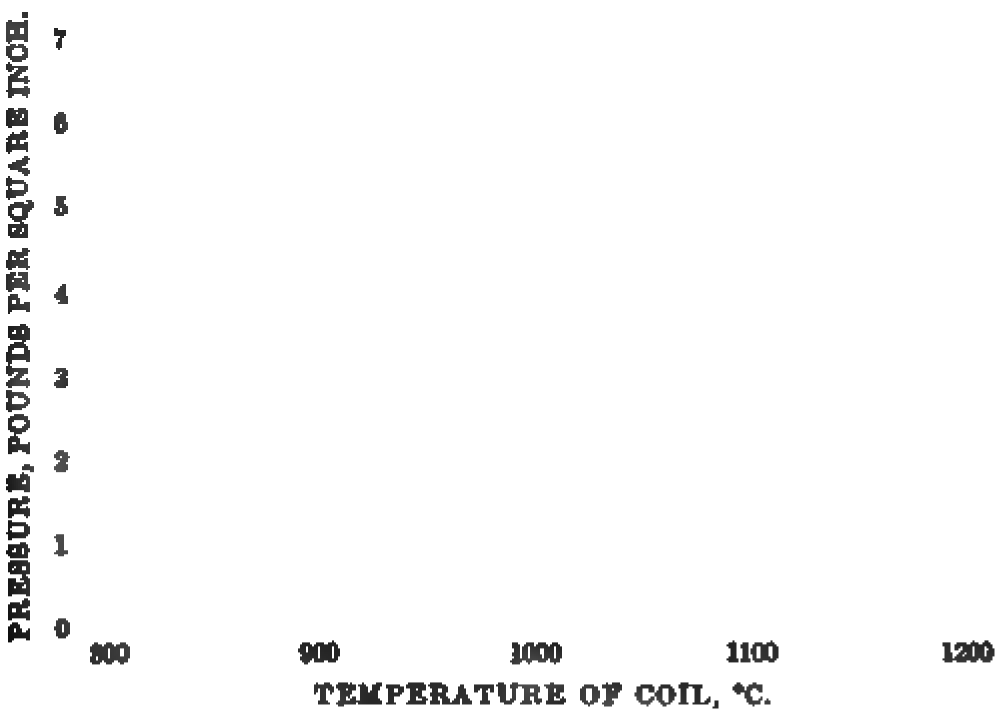

FIGURE 10.—Inflammability curves for all samples from mine No. 13.

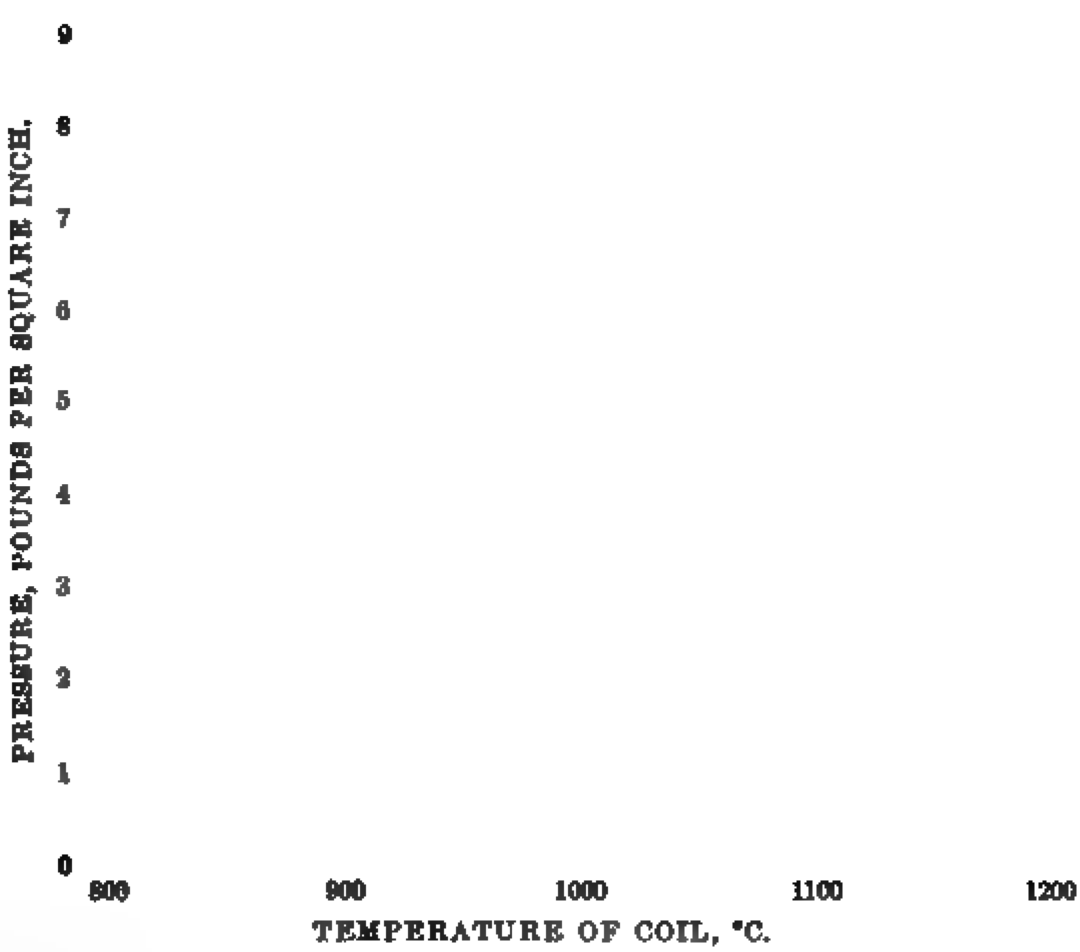

FIGURE 11.—Inflammability curves for all samples from mine No. 26.

measuring device, as it was not expected that dusts that developed so little pressure in the laboratory apparatus would serve to propagate an explosion. At the time the experimental work was done it was

not known that dusts that developed as low a pressure as 0.5 pound per square inch in the laboratory apparatus would under favorable conditions give rise to a dust explosion in the mine.

In the study of the rib dusts it was found that the inflammability of most of the 200-mesh parts of the unground samples was only

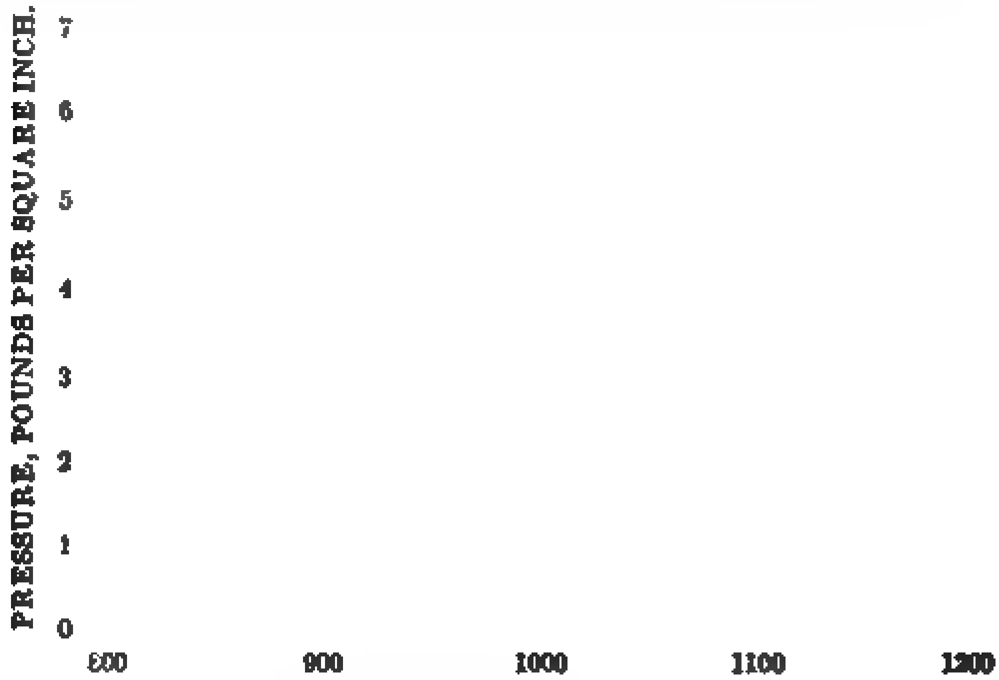

FIGURE 12.—Inflammability curves for all samples from mine No. 49.

slightly less than that of samples ground to 200-mesh size in the laboratory. The question arises: Why does the same relation not obtain for the road dusts? The reason for the difference is not evident. The low inflammability of the sized part of the road dusts

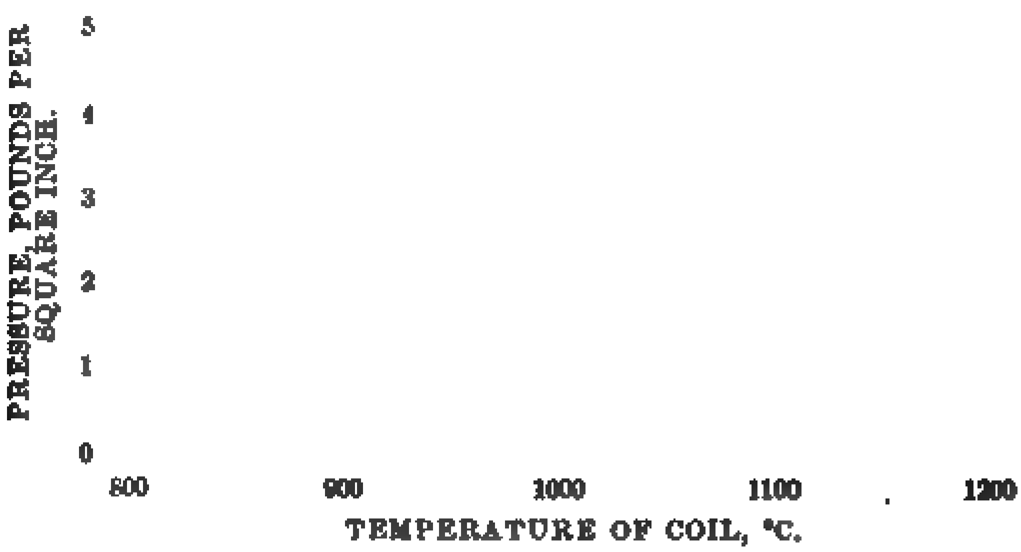

FIGURE 13.—Inflammability curves for all samples from mine No. 60.

may be due to an excess of ash in the finer particles of road dust, though there is no reason for expecting such a selective distribution of ash. A more probable explanation is that the grinding of the dusts in the laboratory produces a larger proportion of extremely fine dust particles than is present in the road dusts of the mine.

No variation of inflammability of the road dusts with the locality in the mine is evident. Among samples whose inflammability is high, as many are from the main entries as from other entries.

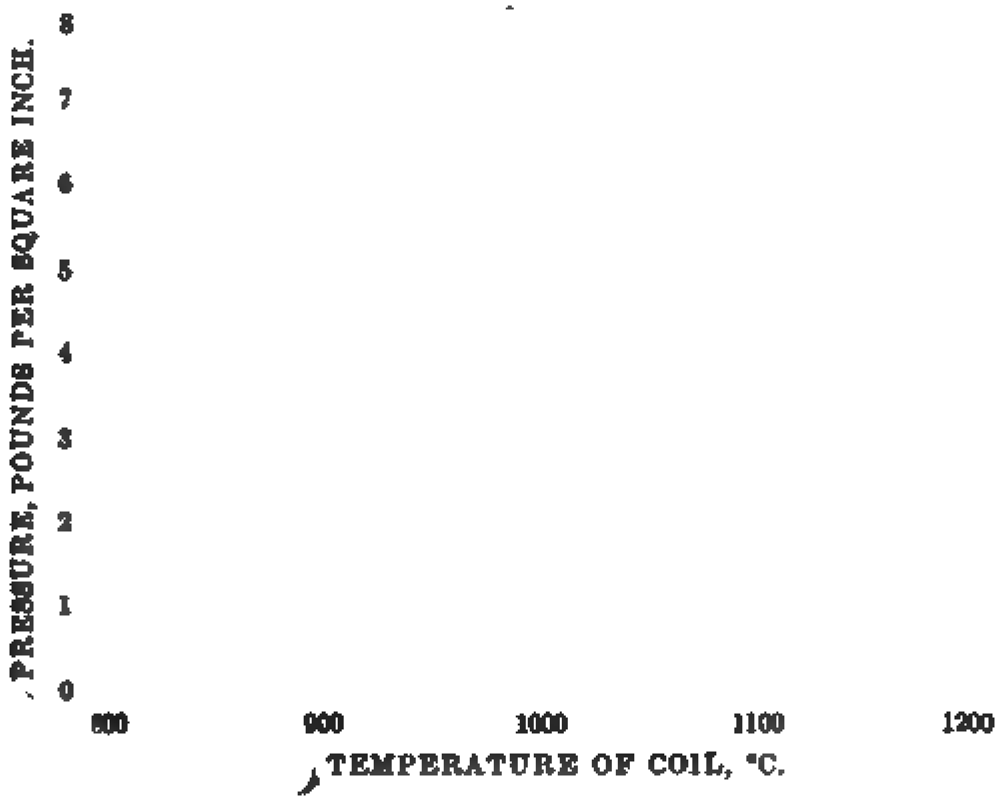

FIGURE 14.—Inflammability curves for all samples from mine No. 88.

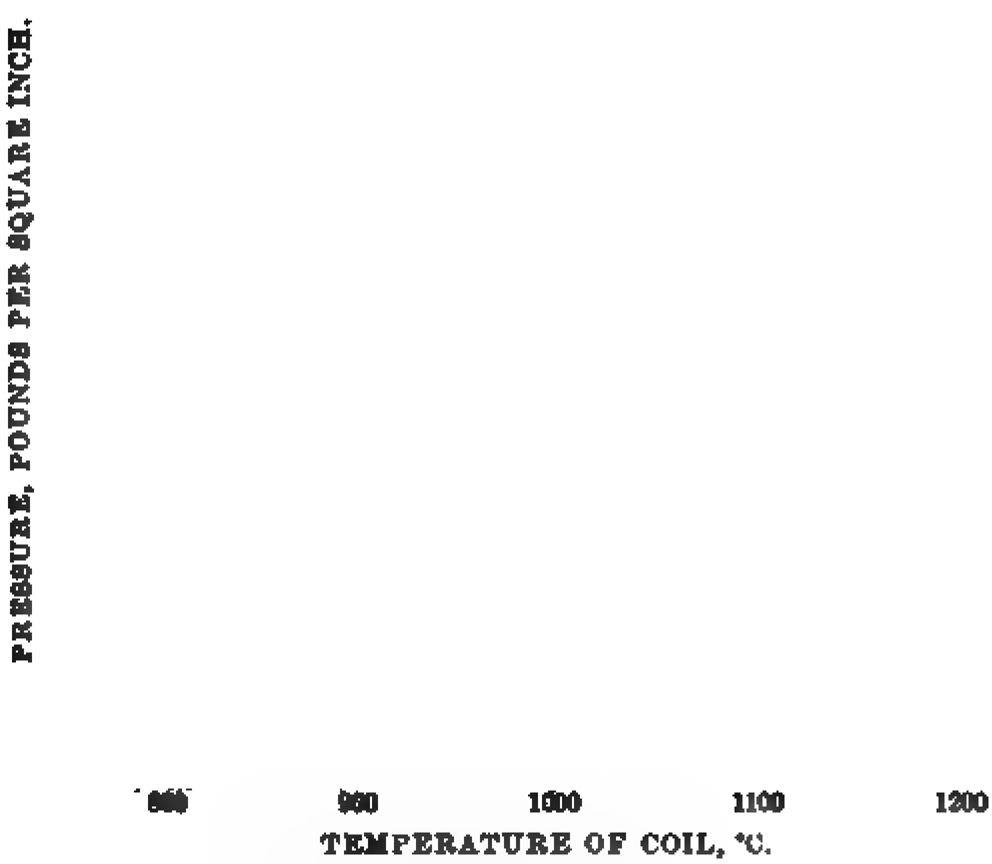

FIGURE 15.—Inflammability curves for all samples from mine No. 95.

If the pressures developed by the 20-mesh and the 200-mesh parts and by the parts of the road dusts ground to pass a 200-mesh screen be compared with the pressures developed by similar parts of the

rib dusts, it will be found that in every case the rib dusts were the more inflammable. And the difference is in the direction one should expect from the difference in the ash content.

Although the laboratory tests demonstrate clearly that in general the road dusts of Illinois mines, when ground to pass a 200-mesh screen, are capable of forming an explosive mixture with air, the results of the tests of the unground dusts are not so conclusive. It would be wrong to conclude from the low pressures developed by the unground samples of road dust that road dusts are not capable of taking part in explosions. Dusts that are too coarse to produce an explosion in the laboratory may propagate an explosion in a mine. In experiments in the large steel explosion gallery of the bureau's Pittsburgh experiment station, dust through a 40-mesh and over a 60-mesh screen propagated an explosion the entire length of the gallery.[a]

To show the relation between the inflammability of the several kinds of samples collected in each mine, inflammability curves for all of the samples from each one of a number of different mines have been drawn on the same sheet. These curves are shown in figures 9 to 15, inclusive.

INFLAMMABILITY OF DUSTS OF EIGHT DISTRICTS.

In the paragraphs immediately preceding, the results of the study of the inflammability of the dusts of Illinois mines have been considered without regard to the geographical situation of the mines. Inasmuch as different beds of coal are worked in different districts of the State, and by different methods of mining, it is reasonable to expect dusts from these districts to differ in inflammability.

To facilitate the comparison of the dusts of the various districts, the more important of the data contained in Tables 1, 2, and 3 have been assembled in Table 4, in which the results are arranged in groups by districts. The pressures reported in the table were all obtained at a coil temperature of 1,200° C.

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[a] See Rice, G. S., and others, The explosibility of coal dust: Bull. 20, Bureau of Mines, p. 43.

TABLE 4.—*Combined data on face, rib-dust, and road-dust samples from mines in various Illinois districts.*

DISTRICT 1.

Mine No.	Source of sample.[a]	Face samples. Proximate analysis (air-dried sample). Moisture.	Face samples. Proximate analysis. Volatile matter.	Face samples. Proximate analysis. Ash.	Face samples. Proximate analysis. Calorific value.	Face samples. Pressure at 1,200° C.	Rib-dust samples. Proximate analysis (air-dried sample). Moisture.	Rib-dust samples. Proximate analysis. Volatile matter.	Rib-dust samples. Proximate analysis. Ash.	Rib-dust samples. Proximate analysis. Calorific value.	Rib-dust samples. Dust through a 200-mesh sieve.	Rib-dust samples. Dust per square foot through a 200-mesh sieve.	Rib-dust samples. Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Rib-dust samples. Pressure at 1,200° C. of— Dust through a 200-mesh sieve.	Rib-dust samples. Pressure at 1,200° C. of— Ground dust through a 200-mesh sieve.	Road-dust samples. Proximate analysis (air-dried sample). Moisture.	Road-dust samples. Proximate analysis. Volatile matter.	Road-dust samples. Proximate analysis. Ash.	Road-dust samples. Proximate analysis. Calorific value.	Road-dust samples. Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Road-dust samples. Pressure at 1,200° C. of— Dust through a 200-mesh sieve.	Road-dust samples. Pressure at 1,200° C. of— Ground dust through a 200-mesh sieve.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
		P. ct.	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*P. ct.*	*Ounces.*	*Lbs. per sq. in.*	*Lbs. per sq. in*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*
1	1	5.20	40.15	9.46	12,078	8.4	2.10	39.80	21.40	10,406	5.47	.0172	0	5.2	8.9	2.63	17.67	68.50	3,425	......	......	.3
	2	......	......	......	......	......	2.63	17.30	73.60	2,455	12.67	.0427	0	0	0	2.23	12.47	80.85	1,285	......	......	0
	3	......	......	......	......	......	2.75	16.30	78.40	1,022	8.31	.0274	0	0	0							
2	1	6.78	40.97	9.47	11,808	8.4	7.80	28.15	59.58	1,438	14.73	......	0	0	0	2.65	20.55	58.53	4,711	0	<.9	.3
	2	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
	3	......	......	......	......	......	......	......	......	......	......	......	......	......	......							
3	1	5.12	41.88	9.68	12,092	8.7	2.10	40.60	22.40	10,771	7.96	.0155	.2	7.9	8.8	3.65	27.85	42.40	6,979	0	<.9	5.2
	2	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......
	3	......	......	......	......	......	......	......	......	......	......	......	......	......	......							
4	1	5.67	39.21	7.19	12,539	8.1	......	......	......	......	1.80	......	0	4.2	8.2	1.33	10.70	85.95	1,055	......	......	.2
	2	......	......	......	......	......	1.28	11.40	85.18	997	41.05	......	0	0	0	2.05	19.85	63.33	4,111	0	<.9	.2
	3	......	......	......	......	......	1.75	10.75	86.03	880	7.10	......	0	0	0							
5	1	6.77	40.63	7.01	12,240	8.0	......	......	......	......	7.25	.0120	.2	1.7	7.3	1.40	13.65	76.60	2,488	......	......	.2
	2	......	......	......	......	......	......	......	......	......	6.67	.0124	0	0	0	1.88	21.92	55.48	5,553	......	......	1.7
	3	......	......	......	......	......	1.98	17.87	69.65	3,337	12.06	.0304	0	0	0							
6	1	6.87	39.98	6.00	12,290	8.1	......	......	......	......	......	......	......	......	......	2.28	22.52	53.43	5,773	0	<.9	2.2
	2	......	......	......	......	......	......	......	......	......	......	......	......	......	......	2.75	29.73	42.45	7,484	......	......	5.9
	3	......	......	......	......	......	......	......	......	......	......	......	......	......	......							
7	1	7.24	40.09	5.65	12,433	8.0	1.55	23.38	48.53	6,664	5.32	......	0	.2	.5	1.85	14.35	75.08	2,223	......	......	.2
	2	......	......	......	......	......	.65	8.68	89.55	585	11.71	......	0	0	0	1.88	14.72	75.48	2,261	......	......	.2
	3	......	......	......	......	......	2.53	14.20	83.45	1,003	8.57	.0189	0	0	0							

[a] In the case of face samples and rib-dust samples, figure 1 indicates that sample was taken from a room; figure 2, that sample was taken from the secondary haulageway; and figure 3, that sample was taken from the main haulageway. In the case of road-dust samples, figure 1 indicates that sample was taken from the secondary haulageway; and figure 2, that sample was taken from the main haulageway.

TABLE 4.—*Combined data on face, rib-dust, and road-dust samples from mines in various Illinois districts*—Continued.

DISTRICT 1—Continued.

Mine No.	Source of sample.	Face samples.					Rib-dust samples.									Road-dust samples.						
		Proximate analysis (air-dried sample).				Pressure at 1,200° C.	Proximate analysis (air-dried sample).				Dust through a 200-mesh sieve.	Dust per square foot through a 200-mesh sieve.	Pressure at 1,200° C. of—			Proximate analysis (air-dried sample).				Pressure at 1,200° C. of—		
		Moisture.	Volatile matter.	Ash.	Calorific value.		Moisture.	Volatile matter.	Ash.	Calorific value.			Dust through a 20-mesh sieve.	Dust through a 200-mesh sieve.	Ground dust through a 200-mesh sieve.	Moisture.	Volatile matter.	Ash.	Calorific value.	Dust through a 20-mesh sieve.	Dust through a 200-mesh sieve.	Ground dust through a 200-mesh sieve.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
		P. ct.	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*P. ct.*	*Ounces.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*
8	1	5.78	40.52	9.10	12,112	8.7	2.65	34.23	28.13	9,641	5.97	....	0	4.9	6.7	1.95	13.05	79.30	1,712	0	<.9	.2
	2	......	......	......	......	......	2.10	13.60	78.78	1,454	21.82	....	0	0	0	2.40	19.30	62.45	3,967	0	<.9	.2
	3	......	......	......	......	......	2.20	15.05	76.15	1,701	17.03	....	0	0	0	......	......	......	......	......	......	......
9	1	4.94	42.06	10.38	11,835	8.9	2.38	41.00	17.80	11,074	7.73	.0255	0	7.0	9.1	2.73	20.32	61.15	3,992	0	<.9	.3
	2	......	......	......	......	......	......	......	......	......	21.94	....	0	.2	4.2	2.65	26.65	47.70	6,167	0	<.9	3.7
	3	......	......	......	......	......	1.60	14.40	75.40	2,237	25.21	....	0	0	0	......	......	......	......	......	......	......
10	1	4.28	42.12	7.83	12,465	8.9	......	......	......	......	.74	....	0	.3	6.6	1.75	15.00	75.83	2,444	0	<.9	.3
	2	......	......	......	......	......	1.85	10.90	83.95	1,208	6.50	....	0	0	0	2.95	23.80	54.13	5,375	0	<.9	2.3
	3	......	......	......	......	......	2.90	24.85	50.33	5,591	39.42	....	0	.2	2.3	......	......	......	......	......	......	......
11	1	4.52	39.68	9.67	12,373	7.9	......	......	......	......	2.63	.0059	0	.2	5.6	2.23	18.57	62.65	4,235	......	......	.2
	2	......	......	......	......	......	1.30	11.85	80.50	1,757	13.03	....	0	0	0	2.18	34.77	45.40	6,966	......	......	4.7
	3	......	......	......	......	......	2.25	21.32	56.93	4,646	64.16	....	0	0	.2	......	......	......	......	......	......	......
Average	......	5.74	40.66	8.31	12,206	8.4	2.32	20.77	63.29	3,943	14.49	....	......	......	2.6	2.27	19.87	63.33	4,110	......	......	1.4

DISTRICT 2.

12	1	3.72	33.53	7.08	12,906	5.4	2.48	32.82	19.60	10,969	5.20	.0203	.2	4.4	7.3	2.15	13.80	69.25	3,377	0	<.9	.2
	2						2.83	29.77	22.58	10,490	15.88		0	4.7	5.6							
	3																					
13	1	4.32	33.68	5.89	13,075	5.4	2.78	32.80	11.90	12,325	13.02	.0178	.2	5.6	7.1	3.68	30.12	16.68	11,232	0	1.4	5.8
	2						4.05	32.15	13.33	11,309	8.16	.0035	.2	2.3	7.1	1.77	12.93	71.19	3,739	.2	<.9	.3
	3						5.23	31.37	45.13	4,444	21.61	.0303	.2	0	.2							
14	1	5.07	34.33	7.26	12,717	5.8	2.50	33.38	19.51	10,721	17.10	.0504	.3	4.7	5.8	3.13	29.07	23.18	10,363	0	1.4	6.2
	2						4.55	31.30	23.95	8,710	3.96	.0004	0	.7	2.7							
	3																					
15	1	4.35	35.20	5.49	13,115	6.6	2.43	32.52	16.74	11,453	8.99	.0093	0	4.3	7.4	2.85	22.40	41.40	7,141	0	<.9	.5
	2						3.50	32.35	17.75	10,870	12.12	.0128	.2	4.4	7.8	2.30	27.60	28.08	9,725	.2	1.4	5.8
	3						4.60	34.13	29.28	7,114	10.39		.3	.5	.3							
16	1	5.77	34.51	6.04	12,856	5.9																
	2																					
	3																					
Average		4.65	34.25	6.35	12,934	5.8	3.50	32.26	21.98	9,841	11.64				5.1	2.65	22.65	41.62	7,596			3.1

DISTRICT 3.

17	1	4.01	42.04	11.15	11,889	8.4	3.45	37.83	24.70	9,887	3.13	.0042	.2	5.8	7.7	7.15	31.30	49.23	4,374	0	0	.5
	2																					
	3																					
18	1	3.98	40.69	10.97	12,013	8.4	3.00	36.85	23.54	10,184	11.33		.2	7.7	7.5	2.78	24.60	58.76	4,835	0	.2	.9
	2						3.90	28.15	50.35	5,396	29.32		.2	.9	1.2							
	3																					
19	1	4.15	42.30	11.33	11,957	8.0	2.80	36.90	20.95	10,667	12.17		.5	6.8	8.0	3.53	22.70	67.68	3,616	0	.2	.3
	2						5.28	35.50	44.40	63.25	3.25		0	1.9	3.1							
	3																					
20	1																					
	2																					
	3																					
21	1	5.43	40.02	8.24	12,388	7.0										2.00	12.00	77.80	2,430	0	<.9	.3
	2															2.90	22.80	49.80	6,194	0	<.9	1.0
	3																					
22	1	4.03	36.90	9.25	12,452	7.0																
	2																					
	3																					
23	1																					
	2																					
	3																					
24	1						2.55	37.00	22.35	10,710	16.63	.0166	.3	7.2	7.3	4.33	26.22	49.08	5,348	0	.3	1.0
	2						5.00	32.13	34.70	7,290	3.79		.3	1.4	1.9							
	3																					
Average		4.32	40.39	10.19	12,140	7.76	3.71	34.91	31.57	8,637	11.37				5.24	3.78	23.27	58.73	4,466			.7

TABLE 4.—*Combined data on face, rib-dust, and road-dust samples from mines in various Illinois districts*—Continued.

DISTRICT 4.

Mine No.	Source of sample.	Face samples.					Rib-dust samples.									Road-dust samples.						
		Proximate analysis (air-dried sample).					Proximate analysis (air-dried sample).						Pressure at 1,200° C. of—			Proximate analysis (air-dried sample).				Pressure at 1,200° C. of—		
		Moisture.	Volatile matter.	Ash.	Calorific value.	Pressure at 1,200° C.	Moisture.	Volatile matter.	Ash.	Calorific value.	Dust through a 200-mesh sieve.	Dust per square foot through a 200-mesh sieve.	Dust through a 20-mesh sieve.	Dust through a 200-mesh sieve.	Ground dust through a 200-mesh sieve.	Moisture.	Volatile matter.	Ash.	Calorific value.	Dust through a 20-mesh sieve.	Dust through a 200-mesh sieve.	Ground dust through a 200-mesh sieve.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
		P. ct.	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*P. ct.*	*Ounces.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. .u.*	*Lbs. per sq. in.*	*Lbs. per n.*	*Lbs. per sq. in.*
25	1	4.76	40.03	12.43	11,723	7.5	3.35	37.10	15.65	11,428	13.15	.0986	.2	7.2	8.4	3.64	24.81	48.21	6,311	0	< .9	2.1
	2						4.23	33.50	21.58	10,121	16.00	.0141	.3	6.8	8.2	2.38	23.37	57.20	5,396	0	1.4	2.9
	3																					
26	1	4.94	39.64	12.89	11,563	8.2	3.20	37.70	20.78	10,651	24.73	.0450	.7	6.5	9.1	3.70	20.45	63.70	3,940	0	< .9	.3
	2						5.80	37.13	27.13	8,962	40.98	.1934	1.4	5.8	6.6	3.20	30.55	39.85	7,763	0	1.4	5.2
	3						4.00	34.65	35.14	7,879	55.48	.1193	3.5	5.8	6.1							
27	1	4.96	39.72	10.72	11,887	8.7	2.75	38.57	21.73	10,530	7.07	.0265	0	5.9	7.9	3.90	21.30	56.05	5,110	0	< .9	.6
	2						4.60	37.15	21.10	10,017	6.57	.0118	.2	5.9	7.7	2.88	26.87	51.20	5,931	0	.4	3.7
	3						5.63	30.87	48.45	5,531	27.63	.0555	.2	.3	.9							
28	1	5.95	39.83	10.87	11,630	8.7	3.00	39.20	17.25	10,823	12.50	.0212	.5	7.3	8.7	3.15	26.40	48.60	6,403	0	< .9	2.5
	2						6.58	34.07	34.10	7,735	8.50	.0240	0	1.7	4.8	2.83	19.22	48.18	6,244	0	< .9	3.3
	3										2.20	.0004	0	0	5.9							
29	1	5.66	38.74	11.98	11,500	7.5	2.43	39.07	19.33	10,939	11.43	.0402	1.7	7.2	8.6	3.15	24.25	56.65	5,049	0	< .9	1.0
	2						4.50	34.60	34.40	7,900	19.09	.0246	.2	.2	5.8	3.00	28.60	44.10	7,205	0	1.4	4.5
	3						6.20	37.63	26.51	8,924	19.12	.0665	.3	6.6	6.6							
30	1	4.45	39.09	12.89	11,633	7.5																
	2																					
	3																					
31	1	5.85	39.65	11.86	11,574	7.3	2.78	40.17	17.60	10,624	5.53	.0362	.2	6.5	8.7	3.70	21.00	56.45	5,026	0	< .9	.3
	2						3.78	37.52	23.53	9,547	27.07	.0179	.5	5.8	7.9							
	3																					
32	1	4.55	40.04	13.20	11,749	7.5	2.10	38.08	19.70	11,147	11.69	.0209	.9	8.9	8.6	5.35	31.10	28.45	8,957	.4	3.9	5.6
	2						5.98	37.62	28.50	8,082	27.68	.1702	.2	3.7	5.3							
	3						9.23	39.17	33.08	6,046	18.15	.1425	0	3.7	.3							

33	1	5.12	38.06	13.50	11,543	8.2	5.15	30.40	21.08	10,237	4.19	.0119	0	4.0	6.5							
	2						7.50	36.78	19.28	9,895	24.07	.0084	.9	3.6	5.9	3.90	27.00	45.20	6,743	0	1.4	2.1
	3						8.00	32.63	29.43	8,077	7.34	.0074	0	.7	1.6							
34	1	4.83	38.92	10.10	11,885	7.9	3.75	37.18	16.30	11,110	9.46	.0232	.2	5.8	7.0	4.83	15.97	73.80	1,939	0	< .9	0
	2						6.45	34.63	24.50	8,996	14.27	.0725	.2	1.7	5.7	4.23	26.02	44.68	6,494	0	< .9	2.8
	3						4.88	32.22	37.55	7,333	13.76	.0105	1.6	2.8	3.6							
35	1																					
	2																					
	3																					
36	1	4.81	39.19	11.56	11,590	7.7	5.40	35.95	18.25	10,570	18.92	.0078	.7	4.5	7.3	3.95	22.05	52.75	4,855	0	< .9	.5
	2						4.33	37.85	16.93	11,005	10.83	.0344	.5	6.5	8.6	5.50	19.80	58.48	3,915	0	< .9	.2
	3						7.55	33.63	27.98	7,992	24.83	.0137	.3	1.6	.9							
37	1	8.08	38.35	11.43	11,090	7.5	4.88	35.40	19.33	10,546	18.56	.0542	.3	5.4	7.0	4.75	28.85	35.50	7,553	.2	< .9	4.4
	2						6.60	33.80	21.78	9,587	24.03	.1692	.3	.5	4.9	4.38	24.72	45.23	6,680	0	1.4	3.0
	3						6.63	32.17	29.53	7,355	25.41	.1804	.2	.2	.5							
38	1	7.58	39.05	11.04	11,315	7.8	4.78	38.50	17.64	10,706	13.50	.0305	.7	6.3	8.0	4.21	19.96	57.99	4,183	0	< .9	.3
	2						7.38	33.05	21.55	9,164	24.36	.2356	.2	3.1	5.4	5.30	31.40	33.25	8,307	0	4.4	5.4
	3						4.28	26.55	42.73	7,022	71.60	.1761	3.8	5.1	3.6							
39	1	7.11	38.07	13.29	11,029	7.2	7.80	37.30	17.07	10,442	15.52	.0464	.7	6.6	7.4	6.30	21.25	55.60	4,149	0	< .9	.3
	2						8.85	32.88	27.33	8,221	22.91	.1407	.2	3.1	5.0	5.03	26.72	46.75	6,338	0	< .9	1.7
	3						8.15	31.80	44.10	4,711	18.38	.1163	1.0	.2	.2							
40	1	5.44	39.11	11.32	11,623	7.4	6.00	36.80	17.78	10,535	6.52	.0209	.5	4.2	6.6	4.80	21.05	51.56	5,051	0	< .9	.5
	2						7.75	31.95	21.00	8,545	35.02	.1597	.3	1.2	.8	3.53	21.62	63.35	3,841	0	< .9	.5
	3						5.78	27.55	45.28	5,540	42.38	.0581	.2	.2	.3							
41	1	6.63	37.20	12.63	11,214	5.9	7.28	33.55	18.63	10,004	19.54	.0193	1.0	4.3	6.1	5.85	23.85	48.08	5,657	0	< .9	.5
	2										5.61	.0252	0	.7	3.3	5.50	26.70	42.25	6,647	.2	< .9	.9
	3						7.10	30.78	53.13	3,724	12.30	.0208	0	0	.2							
42	1	7.69	37.99	9.59	11,497	6.1					6.11	.0015	0	0	.2	4.30	21.50	48.60	6,048	0	> .9	.5
	2										1.51	.0006	0	.2	0	5.33	19.42	57.00	4,563	0	< .9	.5
	3							29.35		5,497	13.38	.0130	0	.2	.3							
Average		5.79	39.03	11.84	11,532	7.6	5.52	35.05	26.20	8,898	18.68				5.1	4.23	24.01	50.30	5,734			1.9

DISTRICT 5.

43	1	3.57	36.48	8.65	12,778	7.7	4.28	31.47	17.98	10,784	12.81	.0356	.2	3.8	6.3	3.40	30.35	19.69	10,431	0	1.4	6.8
	2						7.25	37.13	37.80	5,661	8.14	.0157	.2	.7	.3	1.93	24.82	37.45	8,446	0	1.4	5.4
	3						5.85	30.50	37.53	5,738	12.48	.0842	0	.3	.2							
44	1	3.28	36.43	8.81	12,691	7.3	2.50	34.90	20.48	10,879	11.74	.0124	.5	3.7	7.5	2.25	19.40	52.75	5,656	0	< .9	.5
	2						3.60	32.30	11.78	11,923	16.14	.0642	.7	6.1	8.2	1.55	16.30	61.55	5,116	0	< .9	.4
	3						3.15	27.03	37.28	6,968	14.70	.0254	.2	.2	.5							
45	1	3.41	36.17	8.30	12,706	7.2	3.90	28.40	24.20	9,437	22.91	.0584	.9	.7	.9	2.98	23.57	37.98	7,628	0	< .9	.7
	2						5.28	35.09	30.18	7,000	10.82	.0235	0	.2	.3	3.10	28.05	23.23	10,327	.2	1.4	4.0
	3						4.60	29.90	20.95	10,489	17.43	.1032	.2	3.1	5.2							
46	1	4.35	35.40	9.34	12.411	6.3	3.70	30.43	23.57	10,138	11.12	.0384	.9	3.1	5.1							
	2						4.18	28.37	35.28	7,524	13.29	.0356	.5	.5	.3							
	3						4.40	28.97	27.45	9,443	21.81	.0940	.5	1.0	3.9							
47	1	2.65	36.15	12.82	12,263	6.6	1.80	34.18	19.25	11,196	13.56	.0115	0	5.9	7.0	3.33	20.27	58.25	4,350	0	< .9	.3
	2						4.30	43.22	29.80	5,931	5.78	.0063	0	0	.2							
	3																					

TABLE 4.—*Combined data on face, rib-dust, and road-dust samples from mines in various Illinois districts*—Continued.

DISTRICT 5—Continued.

Mine No.	Source of sample.	Face samples. Proximate analysis (air-dried sample). Moisture.	Face: Volatile matter.	Face: Ash.	Face: Calorific value.	Face: Pressure at 1,200° C.	Rib-dust samples. Proximate analysis (air-dried sample). Moisture.	Rib: Volatile matter.	Rib: Ash.	Rib: Calorific value.	Rib: Dust through a 200-mesh sieve.	Rib: Dust per square foot through a 200-mesh sieve.	Rib: Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Rib: Dust through a 200-mesh sieve.	Rib: Ground dust through a 200-mesh sieve.	Road-dust samples. Proximate analysis (air-dried sample). Moisture.	Road: Volatile matter.	Road: Ash.	Road: Calorific value.	Road: Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Road: Dust through a 200-mesh sieve.	Road: Ground dust through a 200-mesh sieve.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
		P. ct.	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*P. ct.*	*Ounces.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*P. ct.*	*P. ct.*	*P. ct.*	*B. t. u.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*
48	1	3.81	35.50	9.05	12,575	7.2	2.63	32.07	11.60	12,139	16.31	.0210	.5	2.1	7.2	2.63	20.17	49.88	6,329			1.2
	2						3.55	29.40	21.78	9,734	5.99	.0009	.2	.3	6.6							
	3																					
49	1	2.77	38.50	11.40	12,343	7.2					26.56	.0467	0	2.9	5.6	2.5	31.87	24.20	10,089	.4	2.3	6.6
	2						6.20	34.15	30.50	6,961	15.26	.0153	0	.5	.3	2.3	21.32	52.02	6,145	0	<.9	1.9
	3						3.98	25.77	43.65	5,760	43.54	.1071	.2	.2	.3							
Average		3.41	36.38	9.77	12,538	7.1	4.17	31.85	26.73	8,756	15.81				3.7	2.60	23.61	41.70	7,512			2.8

DISTRICT 6.

Mine No.	Source of sample.	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
50	1	6.41	34.47	7.94	12,253	6.3	3.65	27.55	29.85	9,061	32.82	.0190	.7	.3	3.3							
	2						5.15	32.10	25.25	9,144	7.87	.0004	0	.7	.7							
	3						4.33	29.47	24.33	9,794	9.03	.0030	.2	1.2	5.4							
51	1	4.44	35.24	7.70	12,555	5.8	4.31	28.61	21.97	10,300	11.83	.0266			.6							
	2						4.03	38.56	32.26	8,075	19.00	.0775			.6							
	3						4.37	30.17	25.35	9,430	56.26	.1249			.7							
52	1	2.97	39.13	11.21	12,245	8.1	4.54	35.11	14.89	11,426	18.09	.0063			5.8							
	2						3.19	38.61	18.50	11,018	23.29	.0095			6.6							
	3						4.27	32.69	28.68	8,827	17.48	.0192			1.9							
53	1	4.27	35.24	9.01	12,260	5.1	6.48	27.57	16.80	10,526	47.47	.1405			.7	6.64	29.92	17.71	10,726			1.7
	2						5.30	31.28	12.51	11,324	16.44	.0146			.9	5.36	28.24	24.43	9,868			1.0

	3						4.72	26.42	21.09	10,433	33.21	.1694			.9							
54	1	4.28	37.77	10.84	11,934	5.4	5.06	29.54	19.30	10,395	50.30	.0111			1.1							
	2						5.03	33.16	12.24	11,657	14.59	.0095			5.8							
	3						5.45	28.25	30.75	8,474	36.94	.0247			1.5							
55	1	3.48	36.37	10.90	12,154	6.5	4.78	32.24	18.40	9,967	24.59	.0492			.7							
	2						3.23	28.73	23.71	10,093	13.73	.0165			1.0							
	3						4.38	29.79	21.96	9,909	32.49	.0393			.7							
56	1	5.31	35.37	10.77	11,974	5.9	4.99	32.53	15.79	11,176	15.29	.0029			3.7	4.97	23.91	44.05	6,811			.3
	2						5.32	35.35	19.13	9,970	14.55	.0182			.9	5.18	31.32	22.92	10,166			2.1
	3						5.45	31.10	20.82	10,105	18.71	.0116			.8							
57	1	7.05	32.38	9.06	12,049	5.1	4.70	28.69	22.14	10,206	29.77				1.9	4.73	28.65	22.25	10,085			3.1
	2						4.81	30.59	22.07	10,123	28.44				5.8	3.95	26.27	29.17	9,331			4.5
	3										35.10				3.0							
58	1	4.02	37.16	8.35	12,483	6.7	4.05	30.34	24.46	9,857	18.26	.0095			.7							
	2						5.11	36.23	18.20	10,273	21.79	.0122			.9							
	3						4.90	31.42	21.37	10,229	33.99	.0299			1.3							
59	1	7.24	33.03	9.51	11,902	4.9	4.75	31.76	16.29	11,097	45.53	.0911			4.5	5.83	31.20	18.37	10,762	0	< .9	2.3
	2						5.35	29.36	20.50	10,229	50.52	.0354			.9	5.59	28.15	26.92	9,383	0	1.4	1.9
	3						4.32	26.31	41.33	6,651	59.60	.1573			.2							
60	1	3.87	35.37	9.52	12,346	5.4	4.21	31.10	12.32	11,718	22.18	.0073			3.3	7.02	31.42	14.03	10,931	.2	< .9	2.3
	2						5.55	30.91	16.99	10,166	49.16	.0462			.8	4.62	25.97	34.98	8,296			.7
	3						5.39	33.42	24.26	8,469	38.21	.0871			.5							
61	1	6.82	34.88	7.35	12,258	5.6																
	2																					
	3																					
62	1	6.79	33.86	8.08	12,159	4.9	4.42	26.15	37.38	7,864	76.72	.3007			.9							
	2						3.66	33.32	11.81	12,103	37.59	.0376			5.9							
	3						4.14	34.13	13.19	11,693	14.53	.0546			4.5							
63	1	5.39	32.91	9.45	12,222	5.7	3.74	33.30	14.50	11,569	30.74	.0267			5.4	5.33	34.08	16.93	10,895			6.3
	2						6.32	32.40	29.52	8,503	17.21	.0382			1.2	5.43	29.99	22.19	10,134			1.6
	3						6.20	38.11	34.63	7,306	41.18	.1091			.7							
64	1	5.31	37.26	8.73	12,299	7.9	3.13	30.65	12.95	12,004	22.40	.1035			2.6							
	2						5.41	32.76	16.87	9,817	15.08	.0700			.8							
	3																					
65	1	4.18	35.09	8.73	12,478	5.4	4.63	32.32	22.85	10,022	35.99	.0432			5.3	4.60	32.14	23.56	10,174	.5	4.4	5.2
	2						6.42	34.16	16.71	9,997	26.66	.0688			3.0	4.42	32.39	21.79	10,289	.2	< .9	4.2
	3						5.76	29.99	18.66	9,533	35.38	.0736			.9							
Average		5.11	35.35	9.20	12,223	5.9	4.77	31.54	21.46	10,017	29.55				2.2	5.26	29.55	24.24	9,846			2.6

DISTRICT 7.

66	1	7.90	40.20	10.87	11,209	8.0	4.45	38.85	11.43	11,542	11.47	.0253		7.7	8.7	4.05	32.50	28.05	8,467	.2	< .9	5.6
	2															3.13	33.92	23.32	10,049	0	3.9	6.8
	3						5.50	31.76	29.58	7,315	10.48	.0321	0	.2	1.0							
67	1	7.16	39.81	10.92	11,381	7.3	3.95	33.80	20.33	10,375	6.60	.0062	0	1.2	6.7	6.00	28.90	33.85	7,450	0	< .9	3.3
	2						7.33	34.38	35.58	6,754	26.38	.1131	.5	.5	.5	4.80	24.25	47.93	5,499	0	< .9	.5
	3						5.10	25.25	48.80	5,483	15.63	.1036	.2	.7	.5							
68	1	5.98	41.59	9.74	11,646	8.4	3.48	34.97	15.65	11,097	7.68	.0092	.2	1.6	7.9	7.58	34.22	29.65	7,643			2.3
	2						4.83	31.82	30.33	7,664	11.62	.0215	0	0	.9	5.08	30.52	31.50	8,240			5.6
	3																					

TABLE 4.—*Combined data on face, rib-dust, and road-dust samples from mines in various Illinois districts*—Continued.

DISTRICT 7—Continued.

Mine No.	Source of sample.	Face samples. Proximate analysis (air-dried sample). Moisture.	Face: Volatile matter.	Face: Ash.	Face: Calorific value.	Face: Pressure at 1,200° C.	Rib-dust samples. Proximate analysis (air-dried sample). Moisture.	Rib: Volatile matter.	Rib: Ash.	Rib: Calorific value.	Rib: Dust through a 200-mesh sieve.	Rib: Dust per square foot through a 200-mesh sieve.	Rib: Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Rib: Dust through a 200-mesh sieve.	Rib: Ground dust through a 200-mesh sieve.	Road-dust samples. Proximate analysis (air-dried sample). Moisture.	Road: Volatile matter.	Road: Ash.	Road: Calorific value.	Road: Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Road: Dust through a 200-mesh sieve.	Road: Ground dust through a 200-mesh sieve.
1	2	3	4	5	6	7	8	9	10	t11	t12	13	14	15	16	17	18	19	20	21	t22	t23
	P. ct.	P. ct.	P. ct.	P. ct.	B. t. u.	Lbs. per sq. in.	P. ct.	P. ct.	P. ct.	B. t. u.	P. ct.	Ounces.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	P. ct.	P. ct.	P. ct.	B. t. u.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.
69	1	6.71	39.33	12.37	11,232	6.6	3.55	34.48	19.95	10,586	17.67	.0341	.3	4.2	6.8	4.60	20.10	55.73	4,671	0	< .9	.3
	2						5.75	32.33	24.08	8,609	21.86	.0265	1.0	3.2	1.0	4.33	24.87	45.37	6,343	.2	1.4	.9
	3						5.23	24.22	48.33	4,511	26.62	.1248	.2	.2	.3							
70	1	6.44	39.34	12.15	11,374	7.9	3.00	40.75	22.05	10,204	4.59	.0073	.3	5.8	7.9	4.90	27.05	49.03	5,020	0	< .9	.9
	2															4.38	27.52	42.35	6,514	.2	< .9	1.6
	3						10.53	31.57	35.33	5,179			0	.2	.3							
71	1	8.01	40.23	10.96	11,322	6.9	3.13	41.22	16.40	11,221	2.88	.0165	.3	4.5	8.7	4.13	29.02	39.40	6,809	0	6.2	2.1
	2						5.70	37.40	30.90	8,060	8.22		.2	.3	4.9	3.53	35.17	24.33	9,763			7.9
	3																					
72	1	8.95	40.18	10.50	11,164	5.9	3.70	36.50	15.35	11,007	6.27	.0010	0	5.6	9.3	4.70	21.40	59.50	3,913	0	< .9	.2
	2						5.23	36.82	16.80	10,262	5.86	.0158	.3	3.7	8.4	5.30	29.35	31.25	8,024	0	< .9	4.0
	3						6.80	35.38	26.90	7,619	10.05		0	.2	1.0							
73	1	7.93	40.74	9.08	11,527	6.8					2.44	.0044	.2	.3	7.2	5.25	22.80	58.45	4,221	0	< .9	.3
	2						4.25	33.75	19.55	10,253	10.22	.0187	.3	4.7	7.7	4.33	26.72	37.90	7,394	0	< .9	4.5
	3						6.50	35.05	29.08	7,592	12.26	.0254	0	.7	1.7							
74	1	8.44	39.61	10.83	11,122	5.8	3.65	38.90	16.48	10,870	8.23	.0121	.2	7.5	8.0	6.68	30.02	28.95	8,096	0	< .9	3.0
	2						5.50	36.30	40.00	6,309	2.55	.0108	0	.2	.3	7.18	31.07	28.30	7,792	0	< .9	3.8
	3						4.20	35.00	32.05	7,673	3.11	.0053	0	2.1	2.1							
75	1	5.94	40.66	10.16	11,691	7.7	4.85	35.63	17.83	10,508	16.39	.1210	.3	5.1	6.8	4.50	27.35	35.80	7,420	0	< .9	5.0
	2						8.00	33.00	22.60	8,075	28.69	.3032	.2	.2	2.8	3.35	28.10	38.82	7,731	.2	2.3	6.5
	3						6.18	32.42	20.28	9,385	40.73	.3381	.5	3.0	4.5							
76	1	6.22	39.94	11.69	11,534	6.8	4.10	35.78	19.43	10,514	7.31	.0282	.2	4.5	6.6	4.65	12.55	80.25	1,303	0	.9	.2
	2						5.73	32.95	18.13	10,328	11.22	.0188	0	1.2	6.6	4.05	30.35	31.70	8,674	0	1.4	6.3

	3						6.55	30.95	30.75	7,519	4.90	.0054	.2	0	3.0							
77	1						6.15	37.00	12.32	11,045	20.15	.1908	.3	6.6	8.4	4.63	28.62	40.87	6,073	0	< .9	5.2
	2						7.40	38.80	22.43	9,110	10.73	.0166	.3	5.1	6.1	4.23	27.57	38.04	7,472	0	< .9	4.7
	3						6.35	29.10	34.68	7,456	23.65	.0710	.2	0	.7							
78	1	4.40	41.16	12.45	11,666	8.7	3.63	34.59	25.13	9,508	5.42	.0119	0	4.6	7.7	5.03	18.67	59.58	4,054	0	< .9	.3
	2						3.30	33.05	18.93	10,672	9.61	.0191	0	.7	7.3							
	3						5.13	37.07	31.15	6,714	5.20	.0147	0	0	.3							
79	1	4.47	42.34	10.66	11,923	8.6	2.95	34.62	21.45	10,283	10.23	.0097	0	.7	6.8	3.48	22.97	57.90	4,720	0	2.3	.5
	2						3.58	36.57	22.22	9,983	7.10	.0099	.5	5.1	6.6	1.85	11.85	76.09	8,116			.3
	3						5.90	33.20	35.43	6,181	12.20	.0676	.0	.9	.3							
80	1	5.24	38.61	12.41	11,533	8.0	5.65	35.85	19.78	9,988	32.64	.1459	.5	4.2	5.9	5.72	29.58	36.45	7,621			4.0
	2						7.13	38.97	27.60	7,195	5.92	.0140	0	.2	1.0	5.05	32.48	28.25	9.081			6.2
	3																					
81	1	6.64	41.40	10.63	11,687	7.9	3.23	35.62	18.73	10,766	26.39	.0599	.7	5.4	7.9							
	2						4.35	35.80	26.50	8,885	21.30	.0541	.2	.2	3.0							
	3						5.23	30.27	34.25	7,160	18.09	.0304	.2	.2	.3							
82	1	5.28	41.12	11.60	11,700	7.3	2.73	37.70	24.70	9,850	7.81	.0118	.3	1.2	7.5							
	2						4.75	37.93	39.18	5,879	14.75	.0320	.2	.5	.4							
	3						6.25	39.28	39.55	4,990	12.73	.0593	0	0	.3							
83	1	5.81	38.94	12.12	11,399	7.0	6.68	34.27	42.30	5,182	17.67	.0518	0	.2	.3	5.85	29.80	33.08	7,970	0	1.4	4.4
	2						3.05	37.40	23.28	9,896	4.35	.0030	0	4.2	6.8	6.30	26.40	42.10	6,430	0	.9	1.2
	3						6.00	32.75	33.43	7,144	24.91	.1305	.2	.7	.9							
84	1	7.77	38.48	10.67	11,324	7.2	7.43	31.62	37.08	6,777			.2	.2	.7	4.20	30.35	32.67	8,314			5.9
	2																					
	3																					
85	1	8.43	37.53	10.31	11,313	5.9	7.85	32.40	21.43	9,148	40.35	.0783	.3	1.4	1.0	6.66	30.87	22.65	9,364	0	< .9	4.5
	2						8.98	31.75	24.47	8,262	44.23	.1101	.5	1.2	1.1	5.58	30.37	26.68	9,117			6.5
	3						7.10	33.93	29.08	8,267	17.14	.0351	.3	.5	1.2							
86	1	5.80	38.46	11.99	11,552	7.2	5.48	34.32	16.55	10,640	23.84	.0825	.3	5.6	6.2	5.48	34.32	16.55	10,640			
	2															4.25	28.30	35.25	8,231	2	< .9	5.8
	3						5.65	31.85	23.38	9,176	8.29	.0259	.2	.3	1.1							
87	1	6.66	35.67	11.58	11,466	6.8	5.43	37.32	24.65	9,563	4.91	.0055	0	2.8	4.5	5.45	26.00	38.68	6,709	0	< .9	.5
	2						8.23	35.87	23.25	8,354	16.65	.0388	.2	1.2	.7	4.23	32.12	26.69	9,497	0	1.4	5.4
	3																					
88	1	5.83	37.42	12.69	11,317	7.2	5.18	37.09	26.00	8,381	14.45	.0082	.2	1.4	2.3	4.73	35.42	21.25	10,359			6.8
	2						3.03	34.67	24.10	10,006	40.27	.0463	.2	4.7	6.5	4.55	29.73	29.90	8,937			4.7
	3						4.33	33.12	24.28	9,666			.3	4.2	7.0							
89	1	7.91	35.33	9.23	11,723	6.3	6.50	37.55	44.68	4,333	9.71	.0229	0	0	.2	2.93	17.57	65.90	3,757			.2
	2						5.55	32.30	21.30	9,563	10.26	.0097	.3	4.5	4.7	2.12	14.43	72.03	3,002			1.2
	3						4.25	32.55	20.68	10,147	14.98	.0144	.2	4.2	4.9							
90	1	6.40	36.50	10.88	11,479	6.8	4.35	34.85	20.28	10,071	16.52	.0388	.5	5.8	6.3	3.82	27.16	37.19	7,810			4.5
	2						4.55	37.15	16.95	10,712	7.66	.0214	0	5.8	6.6							
	3																					
Average		6.64	39.36	11.10	11,468	7.2	5.31	34.67	26.02	8,710	14.65				4.1	4.71	27.27	40.00	7,075			3.4

TABLE 4.—*Combined data on face, rib-dust, and road-dust samples from mines in various Illinois districts*—Continued.

DISTRICT 8.

Mine No.	Source of sample.	Face samples. Proximate analysis (air-dried sample). Moisture.	Volatile matter.	Ash.	Calorific value.	Pressure at 1,200° C.	Rib-dust samples. Proximate analysis (air-dried sample). Moisture.	Volatile matter.	Ash.	Calorific value.	Dust through a 200-mesh sieve.	Dust per square foot through a 200-mesh sieve.	Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Dust through a 200-mesh sieve.	Ground dust through a 200-mesh sieve.	Road-dust samples. Proximate analysis (air-dried sample). Moisture.	Volatile matter.	Ash.	Calorific value.	Pressure at 1,200° C. of— Dust through a 20-mesh sieve.	Dust through a 200-mesh sieve.	Ground dust through a 200-mesh sieve.
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
	P. ct.	P. ct.	P. ct.	P. ct.	B. t. u.	Lbs. per sq. in.	P. ct.	P. ct.	P. ct.	B. t. u.	P. ct.	Ounces.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	P. ct.	P. ct.	P. ct.	B. t. u.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.
91	1	3.35	38.34	11.04	12,328	8.2	4.33	33.97	13.48	11,543	8.23		.5	1.7	6.8	4.70	25.10	37.78	7,693	0	.2	1.0
	2						5.65	26.05	48.00	6,008	28.58		.3	.3	.3	4.90	24.13	43.10	6,943	0	.2	1.4
	3						6.28	27.12	36.95	7,418	21.72		.3	.3	.5							
92	1	4.42	36.72	12.36	11,939	8.0	6.23	30.87	15.53	11,027	8.86		.2	.3	4.7	4.48	22.65	43.00	6,953	0	.2	1.6
	2						5.65	30.75	15.80	11,068	10.82		.3	.5	3.3	5.00	25.58	38.83	7,306	0	.1	.3
	3						5.58	25.92	48.55	5,555	24.00		.2	.2	.2							
93	1	5.39	38.67	9.33	12,221	8.2	4.63	37.17	11.58	11,774	17.64		.2	3.2	6.3	3.28	18.02	60.18	4,622	0	0	.2
	2						6.33	29.67	35.93	7,571	32.33		.2	.2	.7	2.73	20.10	61.43	4,437	0	0	.3
	3						6.07	29.23	29.85	8,741	19.41		.3	.5	1.0							
94	1	3.12	43.01	10.31	12,517	9.8																
	2																					
	3																					
95	1	5.08	43.13	9.61	12,232	10.1					37.32				6.6	5.15	12.48	78.63	1,093	0	0	0
	2										45.01				5.8							
	3																					
97	1	2.85	41.45	12.19	12,155	9.1					2.63				6.1	4.03	18.25	58.78	4,759	0	0	.2
	2										10.73				7.5	3.55	22.30	48.40	6,235	0	.2	.7
	3										4.39				0							
Average		4.03	40.22	10.81	12,232	8.9	5.64	30.08	28.41	8,967	19.40		.3	.8	3.6	4.20	20.96	52.24	5,627	0	.1	.6

MINES NOT INCLUDED IN DISTRICTS.

90	1	4.03	39.85	8.84	12,476	8.9																
	2																					
	3																					
100	1	3.55	42.30	10.85	12,361	8.2					8.53	.0068	.2	1.9	6.5	2.85	15.05	79.35	1,006	0	<.9	0
	2						5.90	21.85	70.08	1,345	5.83	.0038	0	0	0	3.35	20.85	61.95	3,548	0	<.9	.2
	3							22.30			2.98		.2	.5	.3							
100	1	3.59	38.96	14.76	11,612	8.4																
	2																					
	3																					
Extra	1	2.75	38.68	12.54	12,317	7.9																
	2																					
	3																					
P. S. D.[a]		2.70	36.00	6.30	14,400	9.1																

[a] Pittsburgh standard dust.

DISTRICT 1.

District 1 includes Bureau, Grundy, La Salle, Livingston, Marshall, Putnam, and Will Counties, and part of Woodford County. The No. 2 seam is worked in the district. The coal is highly inflammable, the average pressure, 8.4 pounds, being with one exception the highest developed in the samples from the eight districts. The average volatile-matter content, 40.66 per cent, of the face samples is higher than that of any other district.

All of the so-called "room samples" of rib dust were brushed from the fresh working face, and consisted therefore, probably, for the most part of freshly formed dust. Most of the dusts when ground to pass a 200-mesh sieve proved to be only slightly less inflammable than the face samples, and only one of them developed a pressure of less than 0.5 pound. The 200-mesh parts of each of the original rib-dust samples—that is, the part of the sample that passed a 200-mesh sieve—on the average developed somewhat less pressure than similar samples from other districts. One-half of them developed a pressure of less than 0.5 pound. None of the 20-mesh samples developed a pressure of more than 0.2 pound. Compared with other districts, the unground parts of the room samples of rib dust were decidedly less inflammable. This difference may in part be due to the smaller proportion of fine dust particles in the samples from district 1. The proportion of dust in room samples of rib dust fine enough to pass a 200-mesh screen averaged, in district 1, 5.96 per cent, whereas the average for other districts ranged from 10.8 to 13.4 per cent, and in one district was as high as 31.8 per cent.

As one should expect from the method of mining (long-wall) employed in district 1, the dusts collected in the haulageways there were high in ash, and most of them did not ignite in the inflammability apparatus. None of the 200-mesh portions of the original rib-dust samples from the entries developed a pressure of more than 0.2 pound, two of the ground samples developed relatively high pressures, one ground sample developed a pressure of 0.2 pound, and the remaining samples produced no pressure.

The average ash content of the road-dust samples was about 10 per cent lower than that of the rib dusts from the entries, and a greater proportion of them developed appreciable pressures in the inflammability apparatus. About one-third of the ground road-dust samples developed pressures greater than 0.5 pound. The inflammability of the road dust of this district is doubtless due to coal that falls from the car, as no coal is in place along the entries. A blown-out shot or a gas explosion in a mine would presumably act on both rib dust and road dust, and therefore these dusts must be considered collectively. As, in 50 years of working, an explosion other than a pocket of gas has never been reported from the Illinois long-wall

field, the presumption is that the entries do not contain enough inflammable dust to present the dangers of propagating an explosion.

DISTRICT 2.

District 2 includes Jackson County, where the No. 2 seam is worked. Rib and road dusts were sampled in four mines in the district, and face samples were obtained from five mines. The average content of volatile matter for the five mines was 34.25 per cent, which was lower than that of any other district. The inflammability of the face samples was also the lowest of the eight districts, the average pressure at 1,200° C. being 5.8 pounds.

The samples of rib dusts from the rooms and secondary haulageways of all four mines were highly inflammable. Only two rib dusts from main haulageways were studied and both of these developed low pressures. Of the six road dusts tested, two from secondary haulageways and one from a main entry were highly inflammable; a fourth developed a pressure of 0.5 pound, and two developed pressures less than 0.5 pound.

DISTRICT 3.

District 3 includes Brown, Calhoun, Cass, Fulton, Greene, Hancock, Henry, Jersey, Knox, McDonough, Mercer, Morgan, Rock Island, Schuyler, Scott, and Warren Counties. Seams 1 and 2 are worked in the district.

So far as face samples were concerned, district 3 stood second in the content of volatile matter, the average value for the five mines sampled being 40.39 per cent. The samples of the district were third in relative inflammability.

All of the rib dusts studied, which were limited to samples from the rooms and minor entries, were decidedly inflammable. The road dusts were high in ash and relatively low in inflammability. Out of six samples four developed a pressure of 0.5 pound or more. None of them produced a pressure higher than 1 pound.

Although the rib dusts of this district are unusually inflammable, the road dusts contain a large proportion of inert material and are much less inflammable than those of other parts of the State. As the mines are near the surface, the dusts are wet and therefore in their present condition not dangerous.

DISTRICT 4.

District 4 embraces Dewitt, Fulton, Logan, Macon, Mason, McLean, Menard, Peoria, and Tazewell Counties, and parts of Sangamon and Woodford Counties. No. 5 seam is worked.

The face and the rib-dust samples of district 4 were nearly identical in composition and inflammability with those from district 3. All of the rib dusts from the rooms and secondary entries except those

from mine 42 developed pressures of 0.5 pound or more. With few exceptions the pressures were high. The average pressure for all rib-dust samples, including those from the main entries, was 5.1 pounds.

Most of the road dusts from district 4, unlike those from district 3, were decidedly inflammable. Eighty per cent of the road dusts developed pressures of 0.5 pound or more.

In 1915 an explosion occurred at one of the mines in the district, the flame traveling a distance of 1,000 to 1,200 feet from the origin, being extinguished when it reached the main haulage road.

DISTRICT 5.

District 5 includes Gallatin and Saline Counties, and the district embraces seven mines. Seam No. 5 is worked. According to the tests of the face samples the coal of this district is somewhat below the average of the coal of the State in volatile matter and slightly below the average in inflammability. About two-thirds of the rib dusts developed pressures of 0.5 pound or more. Eighty per cent of the samples of road dusts were found to be inflammable when ground to 200-mesh size. Five out of nine of the 200-mesh parts developed pressures greater than 1 pound.

DISTRICT 6.

District 6 includes Franklin County and parts of Jackson, Perry, and Williamson Counties. Seam 6, east of Duquoin anticline, is worked.

The coal from district 6 is relatively low in inflammability, as well as in the content of volatile matter, the average values being a pressure of 5.9 pounds and a volatile-matter content of 35.35 per cent.

The inflammability of the samples of mine dusts from the district was exceptionally high. Out of 44 samples of rib dust tested, all but one developed a pressure of 0.5 pound and more. Thirteen out of 14 road dusts produced pressures in excess of 0.5 pound. The ash content of the mine dusts was unusually low, the average values being 21.46 per cent for the rib dusts and 24.24 per cent for the road dusts. The practice in the district is to leave the top coal for roof, so that a low ash content in the dusts is to be expected.

The proportion of inflammable rib and road dusts is greater in district 6 than in the mines of any other part of the State. In this connection attention is called to the high proportion of fine dust in the rib dusts from this district. An average of 30 per cent of the samples of rib dust passed through a 200-mesh screen. The proportion of fine dust actually present in the mines is probably greater than 30 per cent, because, on account of the high velocity of the ventilating currents in the district, part of the finest dust was lost in

sampling. That the mine dusts of the district are of a dangerous character is confirmed by the fact that in 1914 an extensive and disastrous explosion occurred in one of the mines of the district.

DISTRICT 7.

District 7 includes Bond, Christian, Clinton, Fayette, Macoupin, Madison, Marion, Montgomery, Moultrie, Perry, Randolph, St. Clair, Shelby, and Washington Counties, and part of Sangamon County. Seam 6, west of the Duquoin anticline, is worked.

The face samples from the district showed about the average inflammability. The content of volatile matter was somewhat higher than the average, and the calorific value somewhat lower than the average for the eight districts. Most of the samples of rib and road dusts developed high pressures. However, a larger proportion of dusts giving a pressure less than 0.5 pound was found in district 7 than in district 6. In district 7, 84 per cent of the rib dusts and practically the same percentage of the road dusts developed a pressure of 0.5 pound or more.

DISTRICT 8.

District 8 includes Edgar and Vermilion Counties. Seams 6 and 7 are worked in the district.

The inflammability of the face samples was higher than that of the samples from any other district, the average pressure being 8.9 pounds. The average content of volatile matter was 40.22 per cent.

Three-fourths of the ground rib dusts developed a pressure of 0.5 pound or more, and most of these developed relatively high pressures, and under favorable conditions would have given rise to explosions. The road-dust samples of the district had an average ash content of 52 per cent, and were less inflammable than those of any other district. Out of nine ground samples four developed a pressure greater than 0.5 pound, and the average pressure for the samples from the district was 0.6 pound. None of the 20-mesh samples developed any measurable pressure, and none of the 200-mesh parts of the unground samples developed a pressure greater than 0.2 pound.

A graphic comparison of the relative inflammability of the dusts from the different districts is given in figures 16 to 22. Figure 16 comprises a map of Illinois on which each mine included in the cooperative investigation is represented by a symbol showing the magnitude of the pressure developed by the face sample with a coil temperature of 1,200° C. Figures 17 to 19 show the pressures developed by the rib dusts, and figures 20 and 21 show the pressures developed by the road dusts.

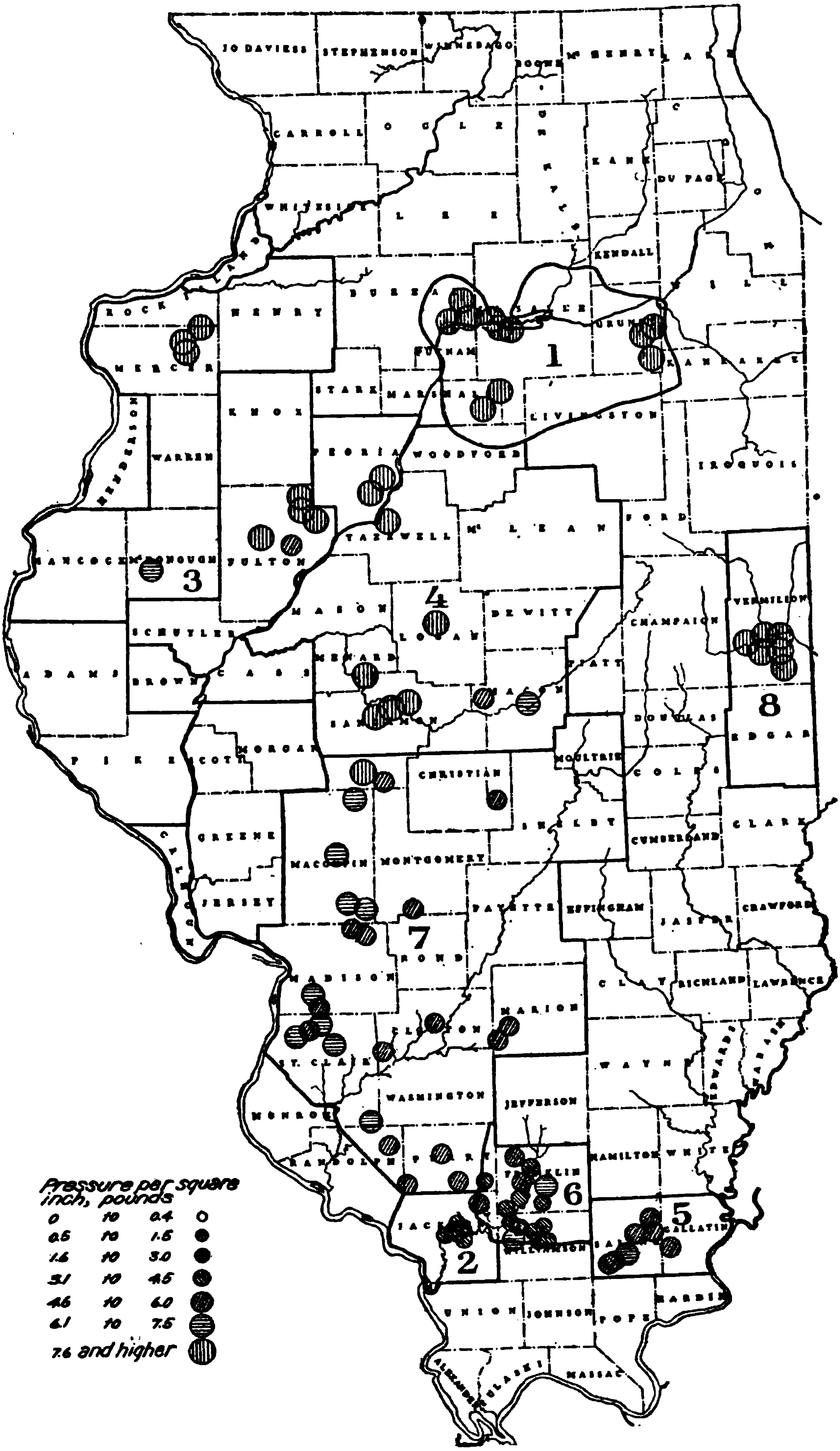

FIGURE 16.—Map of Illinois showing relative inflammability of face samples.

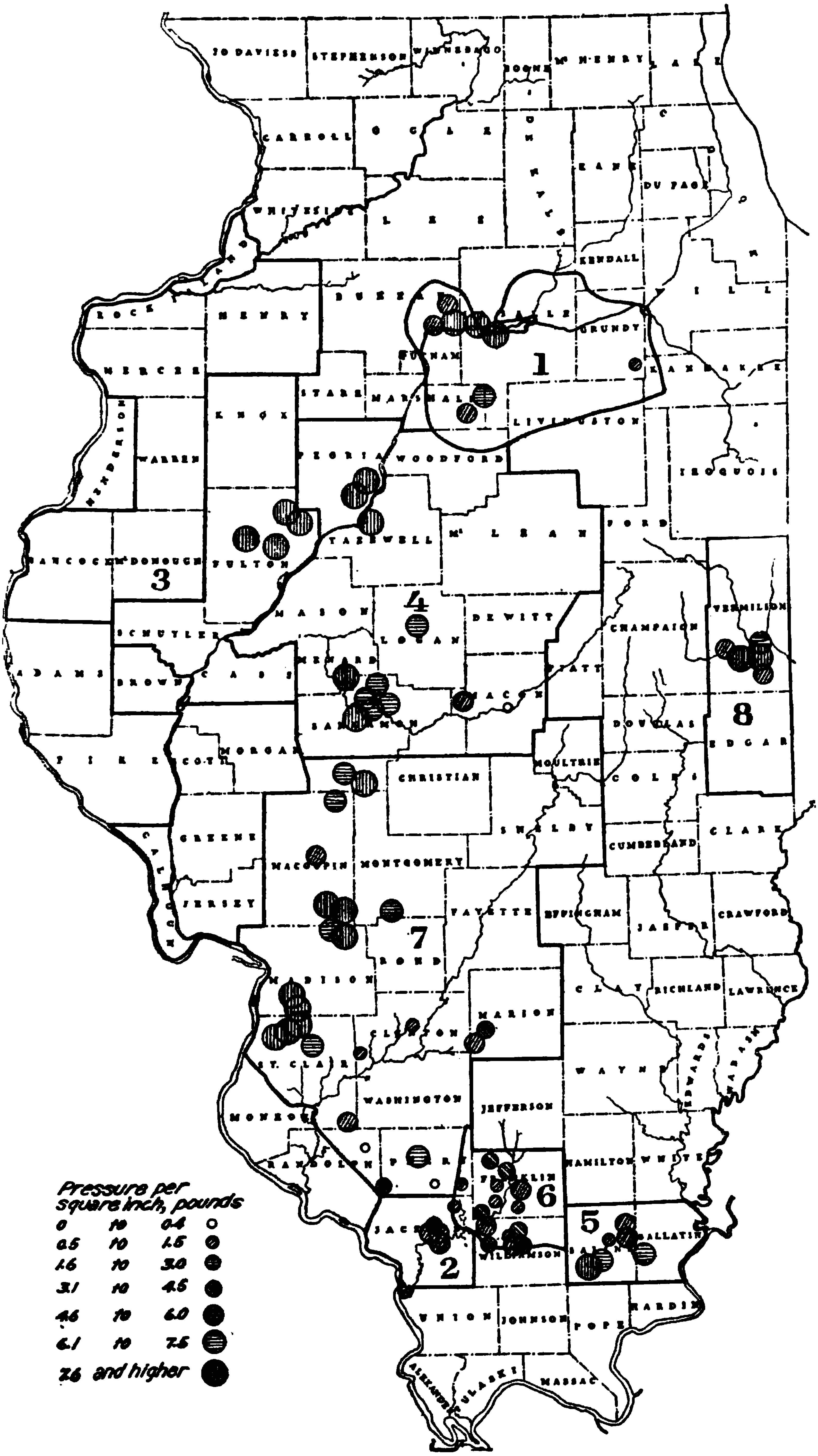

FIGURE 17.—Map of Illinois showing relative inflammability of rib dusts from rooms.

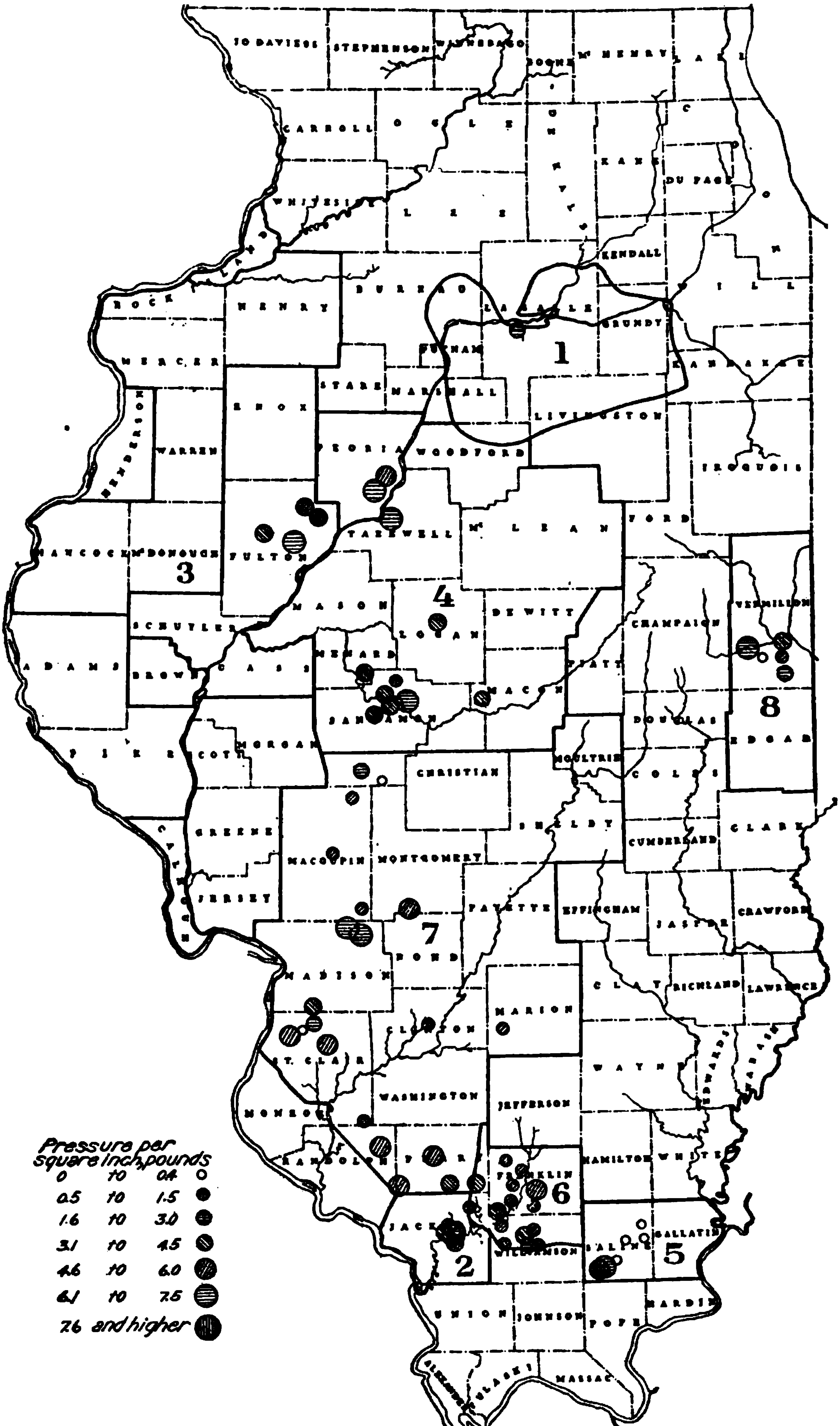

FIGURE 18.—Map of Illinois showing relative inflammability of rib dusts from secondary haulage roads.

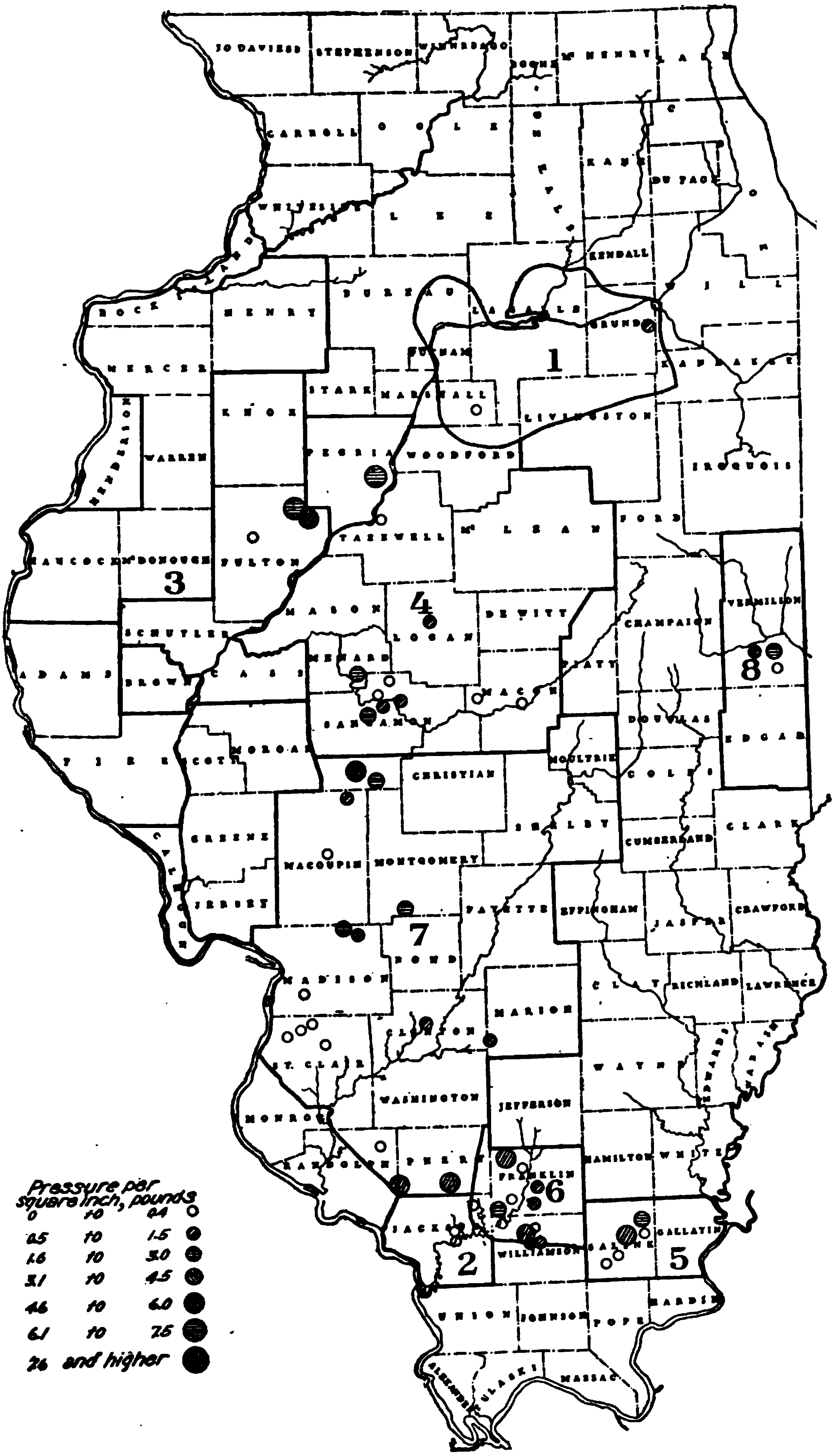

FIGURE 19.—Map of Illinois showing relative inflammability of rib dusts from main haulage roads.

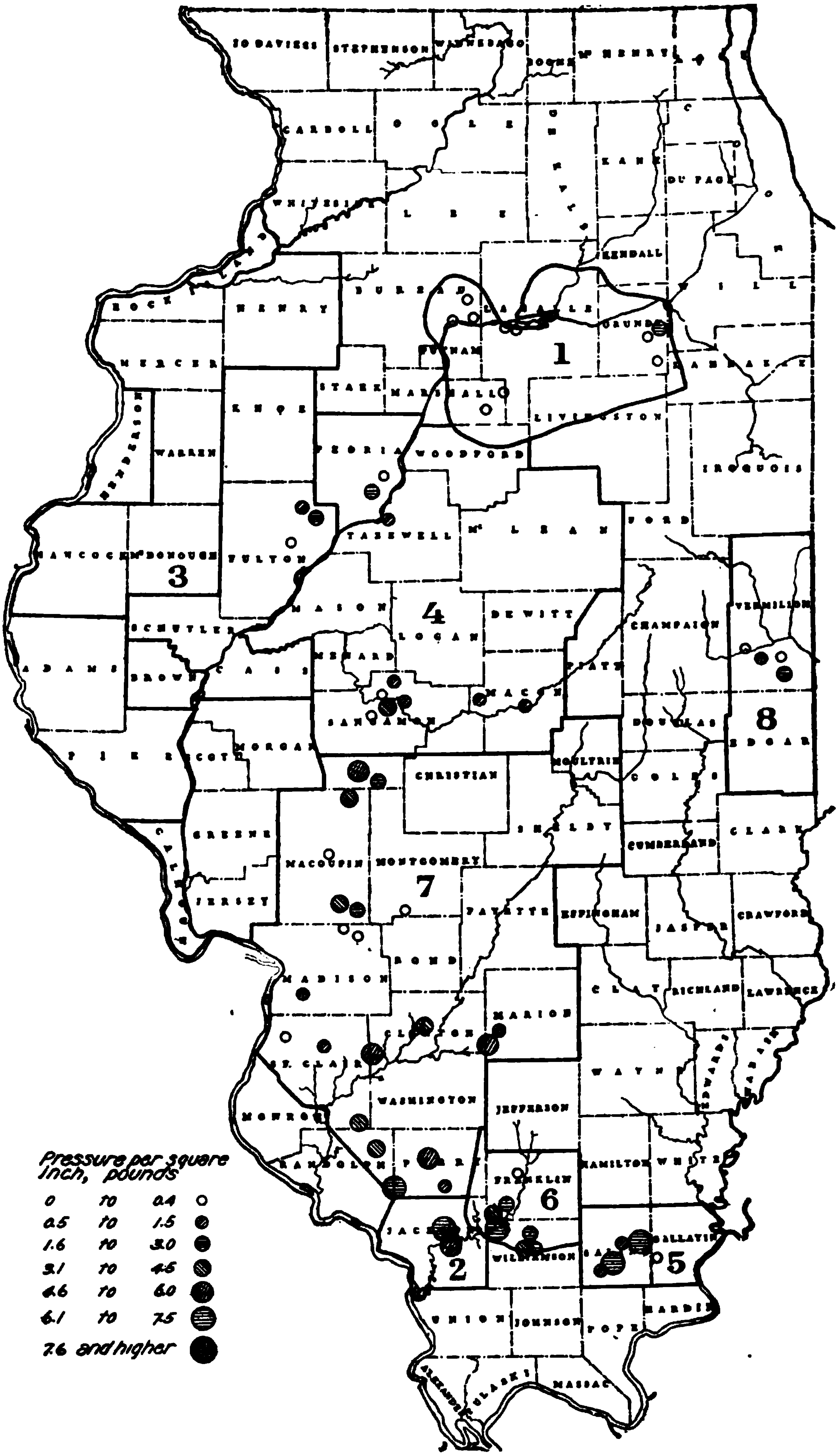

FIGURE 20.—Map of Illinois showing relative inflammability of road dusts from secondary haulage roads.

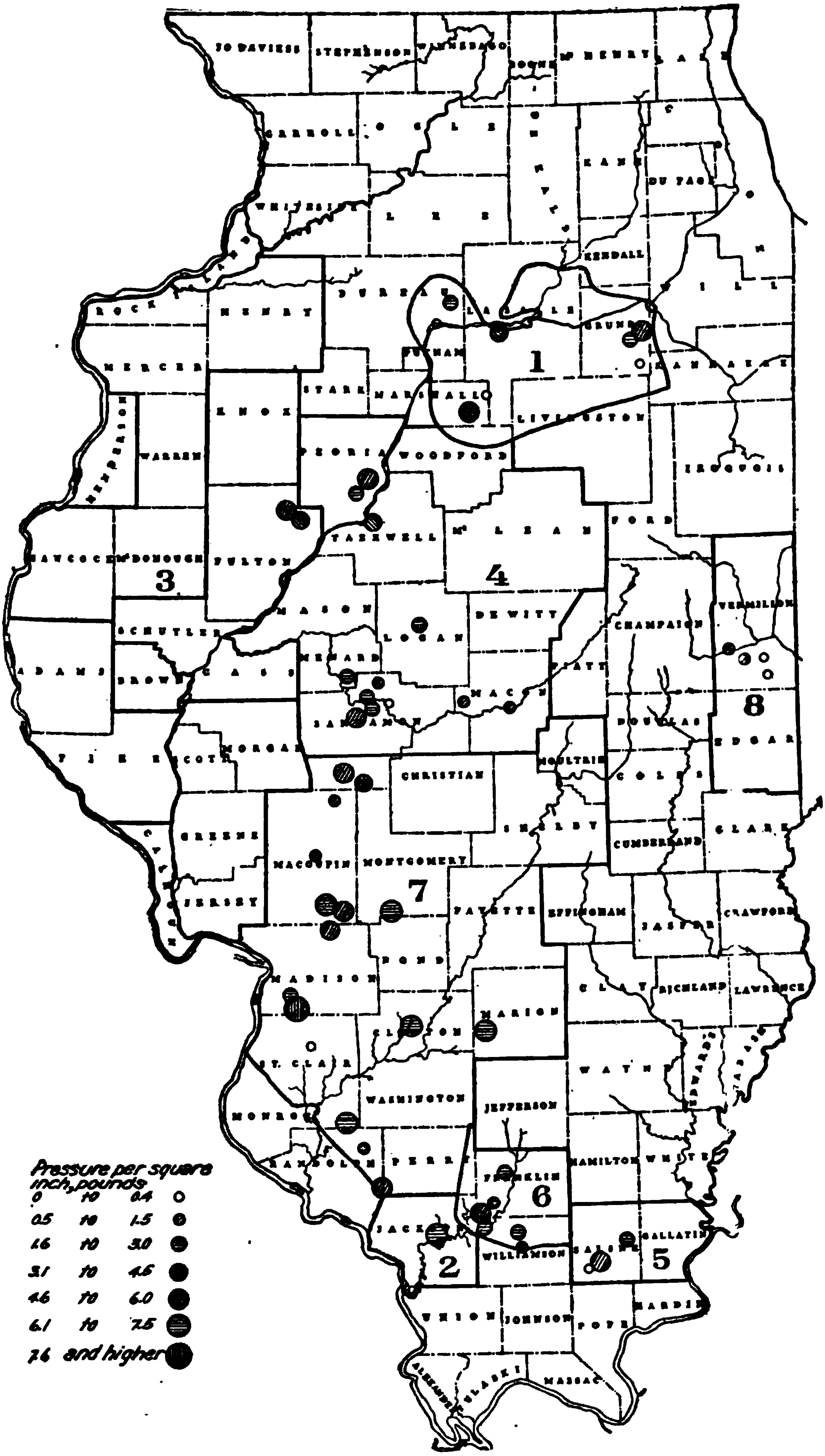

FIGURE 21.—Map of Illinois showing relative inflammability of road dusts from main haulage roads.

RELATION OF INFLAMMABILITY TO CHEMICAL COMPOSITON OF COAL DUST.

The inflammability of coal dust, that is, the readiness with which a cloud of dust inflames, depends on the chemical and physical properties of the dust. For example, the finer the dust particles the more readily inflammable will be the cloud. The greater the content of inert material, as ash and moisture, the lower will be the inflammability. The volatile matter of a given coal has an important bearing on its inflammability. Taffanel,[a] in his gallery tests at Liévin, France, found that the greater the content of volatile matter the more inflammable was the dust. From laboratory tests on a number of coals differing widely in volatile-matter content, the same author[b] found that "the various coals tested, fineness and purity being equal, are approximately classified in the order of their content of volatile matter."

The British explosions in mines committee[c] concluded from laboratory inflammability tests that "the relative inflammability does not depend upon the 'total volatile matter,' but on the relative ease with which inflammable gases are evolved. The order of inflammability so obtained [by their laboratory method] corresponds in a remarkable degree with the percentage of inflammable matter extracted from the same coals by pyridine."

Although the Illinois coals are all of the same type, the data from the unusually large number of samples tested offer an exceptional opportunity for studying the relation between inflammability and chemical composition. In order to determine whether a quantitative relation could be established between inflammability and the volatile matter, the ash, and the calorific values of the face samples, it was assumed that the relation may be expressed by the equation—

$$P = A + B \times \text{volatile matter} - C \times \text{ash} + D \times \text{B. t. u.}$$

in which P is the pressure developed in the inflammability tests with a coil temperature of 1,200° C., and A, B, C, and D are constants. The values of these constants were calculated by the method of least squares from the results obtained for 50 samples selected at random. The agreement between observed values of P and values calculated by means of the equation thus obtained is not satisfactory.

The relation between inflammability and volatile-matter content of the face samples is shown in figure 22. The various symbols used to designate the points show the districts from which the

[a] Taffanel, J., Troisième série d'essais sur les inflammations de poussières; production des coups de poussières, 1910, p. 307.

[b] Taffanel, J., and Durr, A., Cinquième série d'essais sur les inflammations de poussières: Essais d'inflammabilité, 1911, p. 46.

[c] Second Report to the Right Honorable the Secretary of State for the Home Department of the Explosions in Mines Committee, 1912, p. 17.

samples were taken. No definite relation between inflammability and volatile-matter content is shown. There is, however, a decided tendency for the inflammability to increase as the percentage of volatile matter increases.

TESTS OF INFLAMMABILITY OF ROAD DUSTS BY MODIFIED METHOD.

As previously stated, subsequent to the completion of the greater part of the experiments with Illinois dusts it was found that dusts that developed no measurable pressure in the laboratory apparatus were capable of propagating an explosion in the experimental mine of the Bureau of Mines. The laboratory apparatus and method were therefore modified.

Road dusts from district 8 were sampled after the change in the laboratory had been made, and portions of the samples, ground to 200-mesh fineness, were sent to the Pittsburgh laboratory and tested by the new method. The pressure figures obtained by both methods are given in the following table:

Results of inflammability tests of road dusts from district 8, by modified method, samples ground to 200-mesh fineness.

Mine number.	Sample number.	Inflammability pressure by— Regular method. Coil temperature, 1,200° C.	Modified method. Tube temperature, 1,100° C.	Modified method. Tube temperature, 1,200° C.	Group.
		Lbs. per sq. in.	*Lbs. per sq. in.*	*Lbs. per sq. in.*	
92	1	1.6	7.3	....	1
91	2	1.4	7.3	....	1
91	1	1.0	7.4	....	1
97	2	.7	7.6	....	1
97	1	.2	.4	4.3	2
92	2	.3	1.7	....	1
93	2	.3	1.3	....	1
93	1	.2	.4	4.5	2
95	1	.0	.2	.7	3

With the new or "modified" method, dusts may be classified according to their inflammability into three groups. Group 1 includes all dusts that can be ignited by a blown-out shot and give rise to an explosion. Group 2 includes dusts that can not be ignited by a blown-out shot and give rise to an explosion, but may propagate an explosion that has originated in a more inflammable dust or in an inflammable gas mixture. Group 3 includes dusts that will not propagate an explosion.

The numbers in the last column of the table show the groups to which the road dusts of district 8 belong. Six of them would give

rise to an explosion if properly ignited, and all but one would propagate an explosion.

PREVENTIVE MEASURES.

For several years investigations dealing with the prevention and arresting of dust explosions have been conducted under the direction of George S. Rice, chief mining engineer of the Bureau of Mines, and are not yet concluded. As a result of the experiments a number of recommendations have been made regarding the measures to be

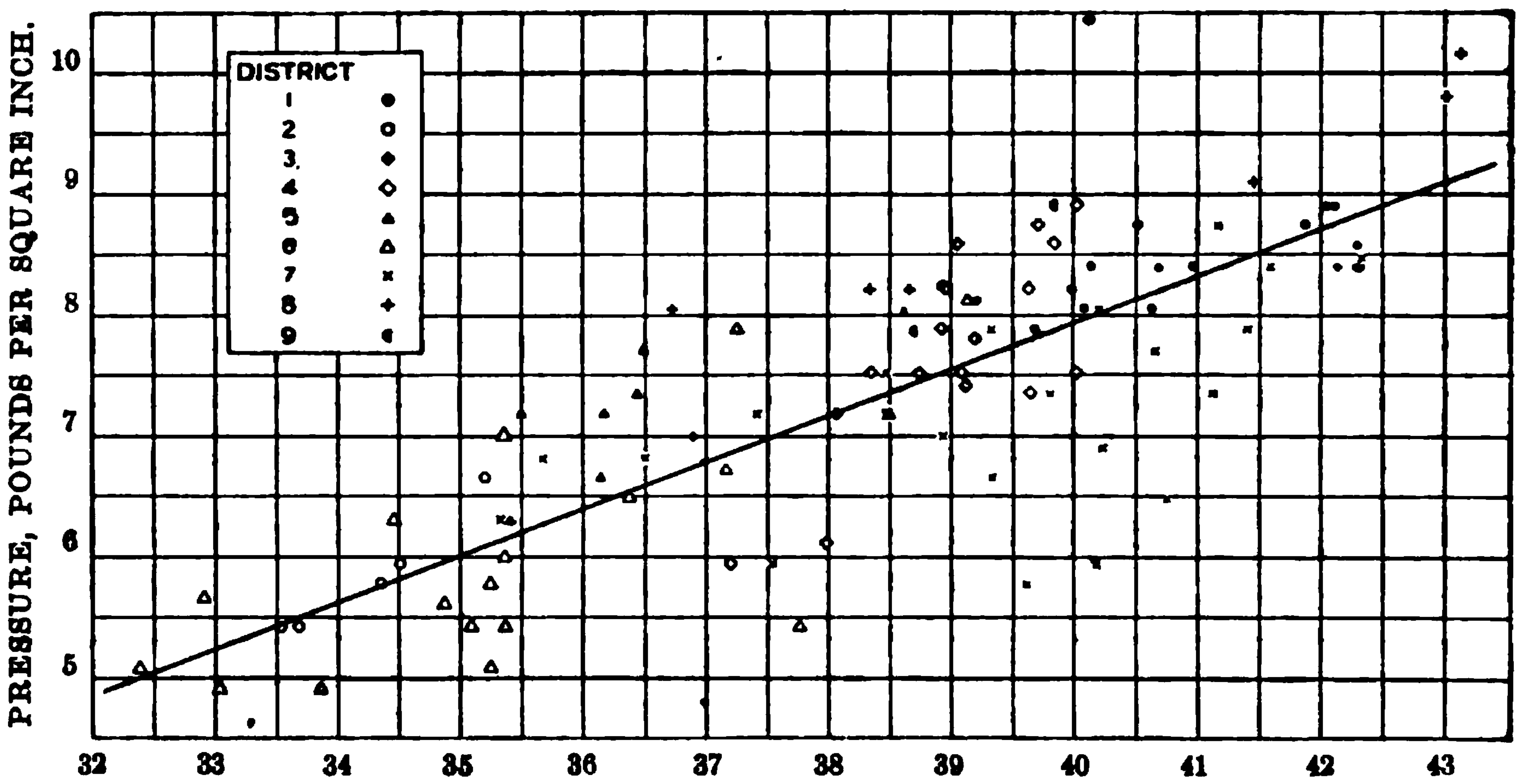

FIGURE 22.—Relation between volatile matter and inflammability of face samples.

adopted for guarding against dust explosions. Some of them have been stated in publications of the bureau [a] and others will be announced as investigations progress.

[a] Rice, G. S., What a miner can do to prevent explosions of gas and of coal dust: Miners' Circular 21, Bureau of Mines, 1915, 24 pp.; Rice, G. S., and Jones, L. M., Methods of preventing and checking explosions in coal mines: Tech. Paper 84, Bureau of Mines, 1915, 50 pp.; Williams, R. Y., The humidity of mine air with special reference to coal mines in Illinois: Bull. 83, Bureau of Mines, 1914, 69 pp.; Rice, G. S., Notes on the prevention of dust and gas explosions in coal mines: Tech. Paper 56, Bureau of Mines, 1913, 24 pp.; Munroe, C. E., and Hall, Clarence, A primer on explosives for coal miners: Bull. 17, Bureau of Mines, 1911, 61 pp.; Rice, G. S., and others, The explosibility of coal dust: Bull. 20, Bureau of Mines, 1911, 204 pp.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF COAL MINING.

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of especial interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45. Sand available for filling mine workings in the northern anthracite coal basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 50. A laboratory study of the inflammability of coal dust, by J. C. W. Frazer, E. J. Hoffman, and L. A. Scholl, jr. 1913. 60 pp., 95 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 56. First series of coal-dust explosion tests in the experimental mine, by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 60. Hydraulic mine filling; its use in the Pennsylvania anthracite fields; a preliminary report, by Charles Enzian. 1913. 77 pp., 3 pls., 12 figs.

BULLETIN 62. National mine-rescue and first-aid conference, Pittsburgh, Pa., September 23–26, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 68. Electric switches for use in gaseous mines, by H. H. Clark and R. W. Crocker. 1913. 40 pp., 6 pls.

BULLETIN 69. Coal-mine accidents in the United States and foreign countries, compiled by F. W. Horton. 1913. 102 pp., 3 pls., 40 figs.

BULLETIN 99. Mine-ventilation stoppings, with special reference to coal mines in Illinois, by R. Y. Williams. 1915. 30 pp., 4 pls., 4 figs.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by M. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 15. An electrolytic method of preventing corrosion of iron and steel, by J. K. Clement and L. V. Walker. 1913. 19 pp., 10 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions, report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 39. The inflammable gases in mine air, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 2 figs.

TECHNICAL PAPER 43. The effect of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp., 10 figs.

TECHNICAL PAPER 58. The action of acid mine water on the insulation of electric conductors; a preliminary report, by H. H. Clark and L. C. Ilsley. 1913. 26 pp., 1 fig.

TECHNICAL PAPER 71. Permissible explosives tested prior to January 1, 1914, by Clarence Hall. 1914. 12 pp.

TECHNICAL PAPER 75. Permissible electric lamps for miners, by H. H. Clark, 1914. 21 pp., 3 figs.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

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Bulletin 103 Mineral Technology 11

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

MINING AND CONCENTRATION OF CARNOTITE ORES

BY

KARL L. KITHIL AND JOHN A. DAVIS

[Prepared under a cooperative agreement with the National Radium Institute]

WASHINGTON
GOVERNMENT PRINTING OFFICE
1917

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First edition. October, 1917.

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PREFACE.

In 1912 the Bureau of Mines had its attention called to the value of the carnotite ore of Colorado and Utah as a source of radium and to the need of investigations being made to prevent waste of low-grade carnotite, to enable the miner to receive a better price for his product, and to insure the profitable recovery of the highest possible proportion of the radium in the ore. The need of comprehensive research was increased by the carnotite deposits being situated on public lands and constituting the largest known supplies of radium ore in the world. Fortunately the bureau was able to interest Dr. James Douglas, of New York City, and Dr. Howard A. Kelly, of Baltimore, Md., who wished to procure a supply of radium for use in the treatment of cancer in hospitals with which they were connected. The suggestion was made that they form a radium institute. Accordingly, the National Radium Institute was incorporated for the purpose of studying the best methods of producing uranium, vanadium, and radium, and the mining and concentration of ores from which these products may be obtained for use in radium therapy.

Subsequently, at the suggestion of the institute, a cooperative agreement was made under which the Bureau of Mines undertook to provide the services of skilled chemists and mineral technologists for the supervision of the mining and concentrating of the carnotite, and the National Radium Institute agreed to furnish the necessary funds for mining and concentrating a sufficient supply of ore and for recovering from it the radium, uranium, and vanadium. Under this agreement a radium-recovery plant was built in Denver, Colo. The plant began commercial operation in June, 1914, and when it stopped work in January, 1917, about 8½ grams of radium had been extracted. The cost of recovery under the methods devised by the Bureau of Mines was only about one-third the current price for radium. In addition, methods of saving the uranium and vanadium in the ores and of obtaining the metals in a pure state were perfected.

The agreement between the National Radium Institute and the Bureau of Mines expired in 1916. As a result of the work under this agreement, the Bureau of Mines has been able not only to demonstrate that radium can be produced at a cost much less than the selling price, but it also has been able to show how losses in mining and concentrating carnotite can be prevented and our resources of this wonderful element conserved. Finally, for an expenditure of less

than $38,000, the bureau has received as its share of the results of the cooperative agreement more than $100,000 worth of radium.

The work of the Denver plant was discussed in Bureau of Mines Bulletin 104.[a] In the present bulletin K. L. Kithil, mineral technologist, and J. A. Davis, assistant mining engineer, describe the methods used in mining and concentrating the carnotite treated at Denver. As the workable carnotite deposits of Colorado and Utah, though lying near the surface, are relatively small, vary greatly in richness, and are extremely irregular in distribution, methods such as are ordinarily used for prospecting and mining shallow deposits have to be modified. Also, it is necessary to examine the ore carefully, sort it closely, and frequently test its radium content with the electroscope. Finally, the carnotite deposits are mostly in an arid region, where water for any purpose is scarce, so that dry rather than wet methods of concentration must be used at most places. Hence the authors have given detailed information in regard to many operations that might receive little notice in an account of the mining of most metals, but the presentation of these facts in regard to carnotite is necessary because of their bearing on profitable mining and concentration, the utilization of low-grade ore, and the prevention of waste.

VAN. H. MANNING,
Director.

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[a] Parsons, C. L., Moore, R. B., Lind, S. C., and Schaefer, O. C., Extraction and recovery of radium, uranium, and vanadium from carnotite. 1915. 124 pp.

MINING AND CONCENTRATION OF CARNOTITE ORES.

By Karl L. Kithil and John A. Davis.

Chapter I. THE MINING OF CARNOTITE ORES BY THE NATIONAL RADIUM INSTITUTE.

DISTRIBUTION AND CHARACTER OF CARNOTITE DEPOSITS.

The principal deposits of carnotite, so far as now known, are confined to a well-defined area lying in the southwestern part of Colorado and the southeastern part of Utah (see Pl. I). Approximately, this area is bounded as follows: On the east by a line extending from a few miles east of the southwest corner of the State of Colorado, to a point about 6 miles west of Naturita, thence directly north to the San Miguel River, thence along the course of this river and the Dolores River to Mesa Creek, thence toward the headwaters of Mesa, Blue, and Calamity Creeks on the south side of the Uncompahgre Mountains; on the north by a line extending from the last-named point westerly to the San Rafael Swell near Green River, Utah; on the west by a line extending from San Rafael Swell to the Henry Mountains, and on the south by a line extending from the Henry Mountains toward the southeast corner of the State of Utah.

This area includes the main districts wherein carnotite has been found, claims have been located, and the ore produced. There are of course, extensive tracts within this area which do not contain carnotite in paying quantities. In places the carnotite beds have been entirely eroded or are not present, or lie so deep that mining is not practicable. The main deposits of carnotite within the area lie in different districts which in Colorado are as follows: Summit and Bush Canyons in Dolores and San Miguel Counties; McIntyre, Gypsum Valley, and Bull Canyon districts; the Paradox Valley, La Sal Creek, Roc Creek, Long Park, Hieroglyphic, Saucer Basin, and Hydraulic districts in Montrose County; Mesa, Blue, and Calamity Canyon districts and the Gateway area in Mesa County. The included districts in Utah are as follows: The Green River and San Rafael Swell district in Garfield County; the Thompson deposits in Grand County;

the Dry Valley, Yellow Wash, and Lisbon Valley districts in San Juan County; the district around the Henry Mountains in Garfield County; and the deposits recently reported in Montezuma Canyon (south of Monticello) in San Juan County.

Outside of the area described there are scattered deposits north of the Denver & Rio Grande Railroad at Coal Creek, near Meeker, Rio Blanco County; along Skull Creek and Red Wash, near Rangely, Routt County; on Horse Mountain, near Eagle, Eagle County, and in South Park, near Garos, in Park County, Colo.

The claims leased from the Crucible Steel Mining & Milling Co. by the National Radium Institute are situated on the north side of East Paradox Valley, in the Long Park and Hydraulic districts. The National Radium Institute has mined no ore except from its leased claims in the Long Park district.

For information regarding the general geology of the region, the probable origin of the ores, and descriptions of other carnotite deposits, the reader is referred to the literature [a] on these subjects.

The carnotite of the area described is deposited in a light-colored, cross-bedded sandstone that lies almost horizontal between the La Plata formation below and the Dakota sandstone (conglomerate) above, and is included in the McElmo formation, which is of Jurassic age. Underlying the white sandstone of the La Plata is the Dolores formation, or "Red Beds," which is plainly visible along and just below the rim rocks of the steep canyon walls in many of the carnotite districts. Along such canyons the ore crops out at different places, so that prospecting is comparatively easy. For ore on the benches and on the mesa prospecting is more difficult, especially where the ore lies at some depth and no surface indications are visible. The carnotite deposits are flat, lenticular, and irregular, varying greatly in extent and thickness. The carnotite itself forms incrustations on exposures of the white sandstone, in joints and fractures of the rock, and is deposited around and between individual sand grains, replacing the original cement. The Utah and Colorado carnotite is amorphous, and to the naked eye appears as a very fine yellow powder. It is often granular, and as a rule can be readily separated from the individual sand grains.

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[a] Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, pp. 9-31. Hillebrand, W. F., and Ransome, F. L., Carnotite and associated minerals in western Colorado: Am. Jour. Sci., ser. 4, vol. 10, 1900, p. 134. Fleck, Herman, and Haldane, W. G., A study of the vanadium and uranium belts of southern Colorado: Report State Bureau of Mines, Colorado, 1905-6, pp. 47-115. Boutwell, J. M., Vanadium and uranium in southeastern Utah: U. S. Geol. Survey Bull. 260, 1906, pp. 200-210. Gale, H. S., Carnotite in Rio Blanco County, Colo.: U. S. Geol. Survey Bull. 315, 1907, pp. 110-117. Gale, H. S., Carnotite and associated minerals in western Routt County: U. S. Geol. Survey Bull. 340, 1908, pp. 257-262. Hess, F. L., Carnotite near Green River, Utah: U. S. Geol. Survey Bull. 530, 1911, pp. 161-164. Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium: Bull. 70, Bureau of Mines, 1913, 100 pp. Hess, F. L., A hypothesis for the origin of the carnotite of Colorado and Utah: Econ. Geol., vol. 9, 1914, pp. 675-688. Kennan, Chester T., On carnotite deposits and the Rand banket: Min. and Sci. Press, vol. 110, 1915, pp. 620-621.

A. STRATUM OF CARNOTITE AND VANADIC SANDSTONE (SEE CROSSES) OF THOMPSONS DEPOSITS, 16 MILES SOUTHEAST OF THOMPSONS, UTAH.

B. OPENING OF NORTH STAR MINE, LONG PARK, COLO., SHOWING CARNOTITE OUTCROP.

MINING CLAIMS OF THE NATIONAL RADIUM INSTITUTE.

LOCATION OF CLAIMS.

All of the claims mined by the National Radium Institute were leased on a royalty basis from the Crucible Steel Mining & Milling Co., a subsidiary company of the Crucible Steel Co. of America. Of the claims leased, 11 were in or near Long Park, Montrose County, Colo., 6 were in the Hydraulic district, Montrose County, and 10 were in the vicinity of Sawpit, a station on the Rio Grande Southern Railway, a few miles west of Telluride, San Miguel County. The last-named claims contain no carnotite, but vanadium solely. Of the carnotite claims, only those in the Long Park group were mined, all the ore required being obtained there.

Long Park is the name given to a plateau lying north of East Paradox Valley. This plateau is roughly elliptical in shape, being about three and a half miles long and perhaps a mile or a mile and a half across at its widest part. The park slopes gently northeast to the base of an escarpment, which is capped by Dakota sandstone, and pierced by three canyons that carry the drainage from the park to the San Miguel River. Toward the southwest the park rises gradually for about half a mile to the crest of the rim of East Paradox Valley.

The camp of the National Radium Institute is in a small box canyon at the extreme western end of the park. It is 58 miles from Placerville, the nearest railroad station, and 15¾ miles from Naturita, the nearest town and post office on the road to Placerville. The camp buildings and the spring are on the Maggie C mill site, a rectangular piece of ground comprising about 5 acres; it and the Maggie C lode are the only patented claims in the vicinity. A view of the camp is shown in Plate VI, *A* (p. 48).

The Vanadite and Uranite claims are nearest the camp. The Vanadite adjoins the mill site, as shown on the map (fig. 1), and embraces the crest and rim of a ridge, or hogback, about 150 feet high, capped by the sandstone that usually carries the carnotite. This ridge forms the north wall of the canyon in which the camp lies. The Uranite claim crosses the end of the Vanadite claim, and extends down the northern slope of the hogback. Overlapping the end of the Uranite and roughly parallel to the Vanadite claim is the Henry Clay, which lies wholly on the north slope of the ridge.

Of all the claims in Long Park the Maggie C is the most important, as it has been by far the best producer. The southern end of this claim is on the level floor of the park, one-quarter of a mile east of the mill site (see fig. 1). About 300 feet from its southern end the claim crosses a continuation, about 25 or 30 feet high, of the ridge on which the Vanadite, Uranite, and Henry Clay claims are located. From

the crest of the ridge the ground slopes gradually for about 500 feet to a gully, dry except in storms, running nearly at right angles with the axis of the claim; on the other side of the gully the ground rises over another low ridge and then slopes downward to the northern end of the claim.

The Florence, Nellie, and Great Western claims (see fig. 1) are not in the park itself, but on the side of a hill in the drainage area of Hieroglyphic Canyon. The Medea claim lies on a low spur projecting into the southern edge of the park about 2½ miles from the camp. The Dixie claim is in the rim rocks of East Paradox Valley just east of the extreme eastern end of the park; and the Buckeye and Noontime claims are in the rim rocks, about 5 miles from the camp, along the road to Naturita.

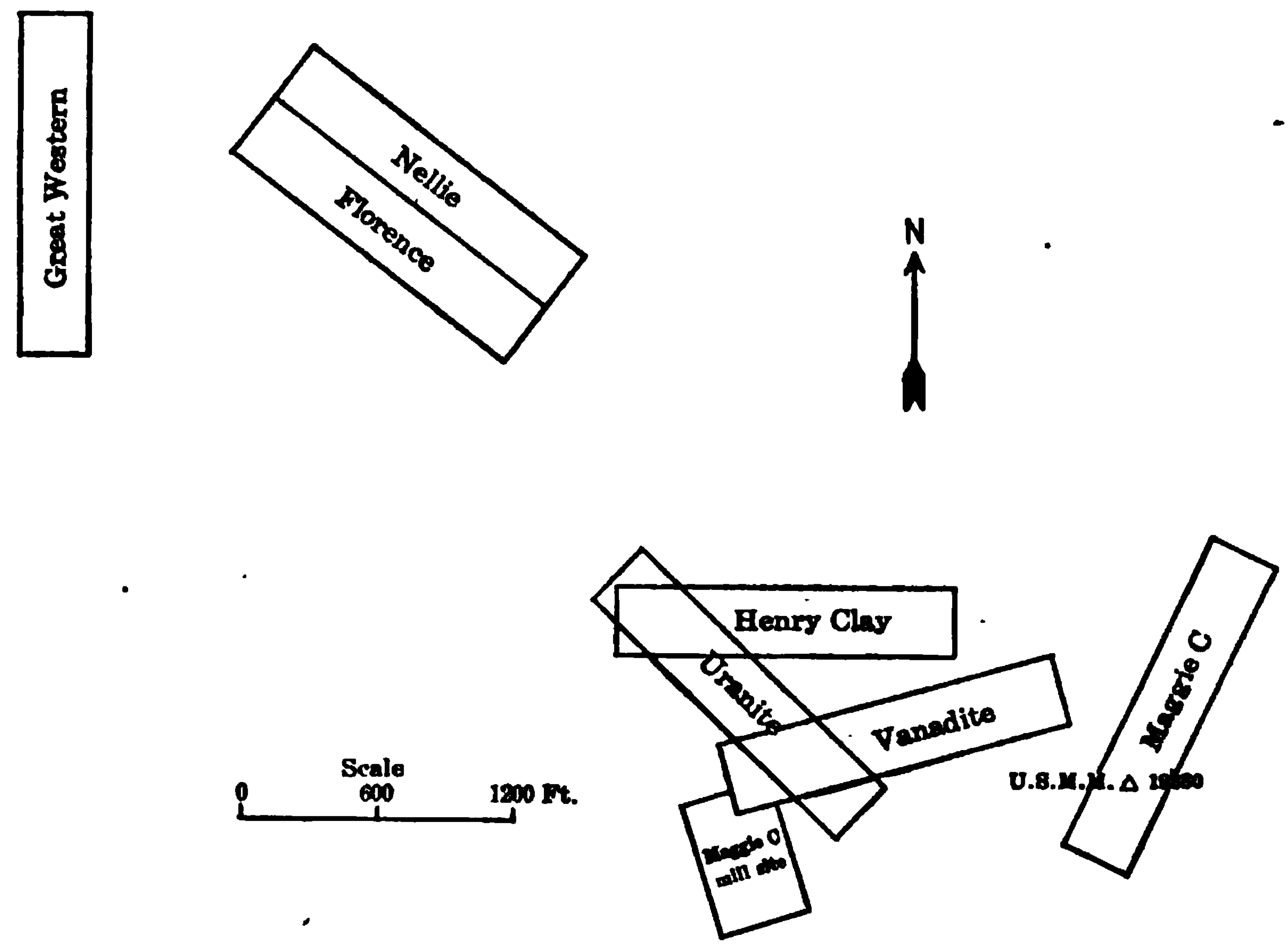
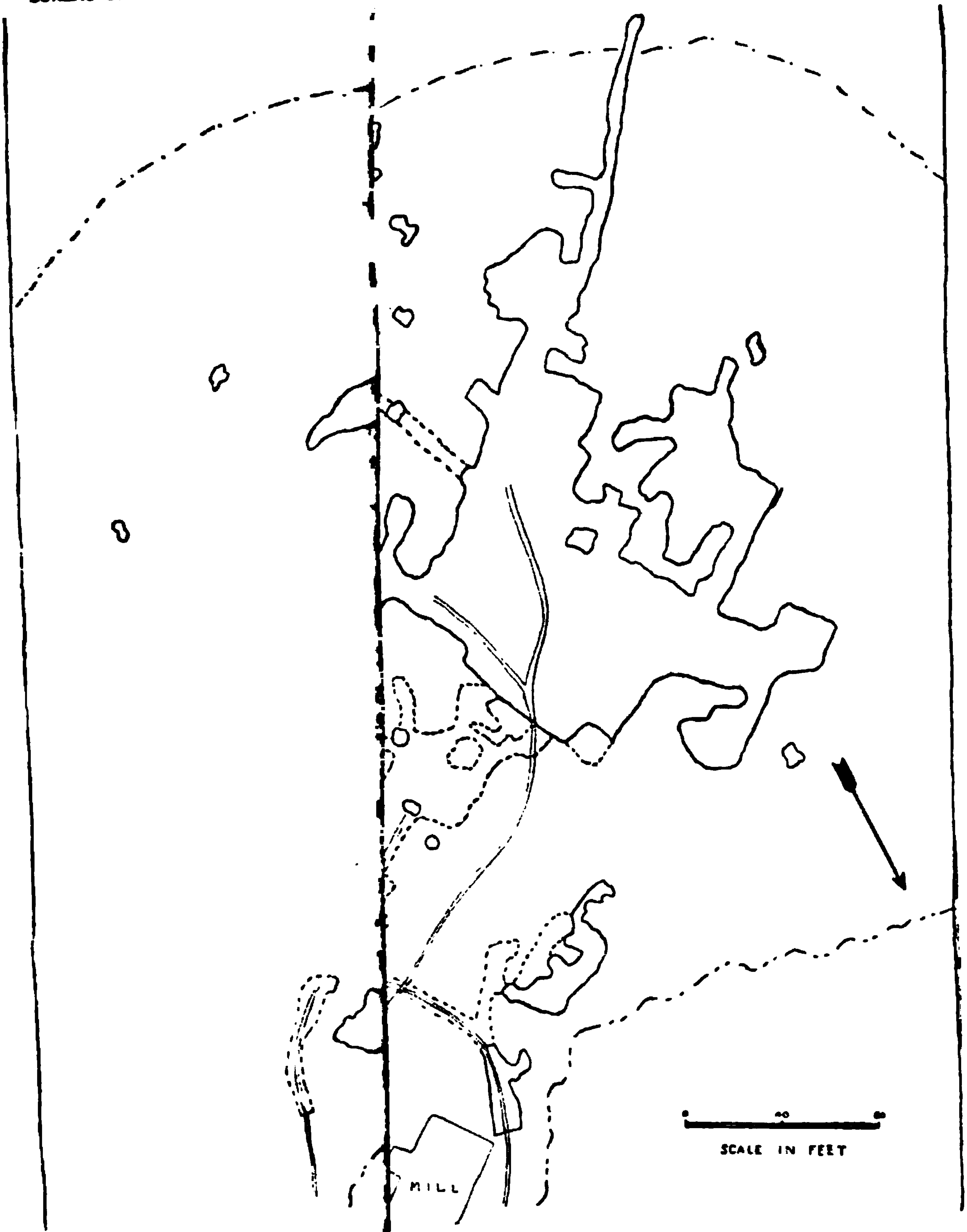


FIGURE 1.—Sketch map o some of the claims worked by the National Radium Institute.

DESCRIPTION OF THE CLAIMS.

MAGGIE C.

The ore on the Maggie C claim was found entirely within the area between the low ridge and the gulley described above, and at a fairly definite geologic horizon embracing 10 to 15 feet of sandstone. The main body of the ore comprised an irregular but rather elliptical pocket measuring about 350 feet long by 200 feet wide at the widest part, with its main axis trending northeast. Within this pocket there were three places near which the ore was more abundant and of better grade than elsewhere. The largest of these rich spots occupied most of the southern half of the pocket, and is indicated by the large open


MILL
40
SCALE IN FEET

cut near the center of map 2 in Plate II. This cut included most of the productive part of this pocket, although, as shown on map 3 (Pl. II), the southern limits of the cut were later somewhat extended. The second rich spot, although rather small, was remarkable for the thickness of the ore, for in places solid shipping ore 3 and 4 feet thick extended across the entire face of the development tunnels.

This area is best shown on map 3 (Pl. II), where it is indicated by the dotted lines (representing the underground work), east of the main open cut. The narrow open cut connecting this pocket with the first contained only milling ore, and very little of that. The third large pocket, which was mined by tunneling from the gulley, is also outlined in map 3 (Pl. II). Besides the three large pockets, three smaller ones were found on the Maggie C (see Pl. II), two along the rim rock south of the main lens and one near the entrance to the main tunnel.

Plate II clearly shows the progress of development and mining. Map 1 indicates the amount of work that had been done on this claim before it was leased to the National Radium Institute. In the prospect pits shown, ore was found within a few feet of the surface, except at ×. Map 2 shows the development in February, 1915. Map 3 shows the condition of work on October 15, when the mining of shipping ore for the year 1915 ceased. All of the pillars of the tunnel had then been drawn, including those that carried milling ore only, and the pocket had been connected with the two other large pockets. The position of the shafts or raises for light and air is indicated on the maps.

The production of ore from these various pockets was approximately as follows: Main open cut, 265 tons; large stope, 150 tons; tunnel, 210 tons; small pockets on rim, 8 tons; small pocket on gulley, 10 tons; other underground work indicated on map 3, 7 tons.

Plate III shows a vein in the open-cut workings on the Maggie C.

FLORENCE.

On the Florence claim the main ore body found was in the side of a small gulley that crosses the claim about 300 feet from its eastern end. Along the sides of this gulley, which is dry except in storms, the carnotite-bearing sandstone outcrops; but the ore is pockety and varies greatly in character. On the east side of the gulley, near the north line of the claim, a "bughole" 3 to 5 inches in diameter contained some high-grade carnotite. This ore was so moist and soft when first opened that a miner's spoon could be pushed to its full length into the hole without difficulty. On exposure to the air and drying this high-grade ore became somewhat harder, but could still be crumbled readily with the fingers. Much of the

rock surrounding this "bughole" was so impregnated with carnotite for 18 inches or 2 feet as to make it shipping ore; beyond this the sandstone usually showed traces of uranium and vanadium, but was too low grade to be milling ore. A tunnel driven along this hole for 60 to 75 feet, obtained just about enough ore to pay expenses, but found no considerable pocket. Farther up the gulch and on the same side, ore of entirely different appearance was found as thin bands or laminations in a clearly bedded sandstone. The bands of carnotite were one thirty-second of an inch to 5 or 6 inches across, the extremely thin bands being nearly pure carnotite and the thicker ones being sandstone so impregnated with the carnotite as to carry 2½ to 3½ per cent U_3O_8. The sandstone between the bands of carnotite was in many places highly impregnated with vanadium, the whole forming a mineralized band or vein 2 or 3 inches to 2 feet thick. On the west side of the gulley, about halfway between the center line and the north line of the claim there was a "bughole" similar to that on the east side but not as large; it was followed by a tunnel for about 25 feet, when the ore gradually became laminated and banded, and widened so that the stope measured approximately 20 by 30 feet. At the far end of this stope the ore sloped upward sharply and was exposed in the creek bed. On the mesa west of the gulley smaller pockets, similar to those on the Nellie claim, were found which yielded a few sacks to 3 or 4 tons each, but a large proportion of the ore from the Florence was mined underground, especially in the tunnel and stope.

The Florence claim ranked next to the Maggie C in productiveness, yielding about 140 tons. Work on the claim was continuous for 16 months. Much of the ore was excellent and the total output averaged, by electroscopic tests, 2.90 per cent U_3O_8

NELLIE.

On the Nellie claim, which ranked fourth in productiveness, there were two main pockets, each of which produced over 10 tons of shipping ore. Both pockets were near the surface of the mesa, the average thickness of the overburden removed being only about 3 feet. The first pocket was at nearly the middle of the claim, and the other near the middle of the east half. The ore from these pockets was rather above the average, rarely containing less than 3 per cent U_3O_8. Extensive prospecting revealed several other small pockets which produced a few sacks to a ton or so each, but no other large paying pockets, although the many small outcrops may indicate that more extensive development might prove profitable. Work was carried on continuously during 16 months and the total output was about 42 tons, which averaged higher in uranium than that from any of the other claims worked, the electroscopic tests showing 3.65 per cent U_3O_8.

SURFACE WORKINGS ON MAGG E C CLA.M

GREAT WESTERN.

On the Great Western claim ore was mined at three main workings, the first lying at about the middle of the north half of the claim, on the north side of the steep-sided gulley mentioned, the second being south of the center of the claim, on the south side of this gulley, and the third near the middle of the south half of the claim but higher up the hillside than the second.

In the first locality, the ore formed irregular impregnations in fairly massive sandstone. Most of the ore was near the outcrop and was mined by open cut; two tunnels, about 50 feet and 30 feet in length, followed more or less well-defined leads. Several other small pockets were found here in the sandstone outcrop along the rim of the gulley; they yielded a few sacks each, and the rock between was entirely barren. The total production from this locality was approximately 30 tons of shipping ore and a small quantity of milling ore.

In the second locality ore was found in one large pocket and in two smaller ones that were opened in prospecting and mined through short tunnels. The main pocket, which yielded about 30 tons of shipping ore, was also discovered by surface prospecting, but a crosscut about 25 feet long was driven from the gulch to facilitate mining. At the third locality the ore was found in a loose breccia, different sized pieces of carnotite-bearing sandstone being indiscriminately mixed with sand, clay, and angular fragments of barren rock. The mining here was usually by small open cuts that followed the more finely broken material around the larger bowlders, many of which weighed more than 10 tons. In the winter, when the ground was frozen and snow-covered, the ore was followed by small tunnels or drifts with only 2 or 3 feet of cover. The ore was usually good, and numerous pieces carried as much as 6 and 8 per cent U_3O_8. In spite of the irregular distribution of the ore, which necessitated constant prospecting, the claim was worked with profit during the 17 months of active mining, and ranked third in yield. The uranium content of the ore averaged better than 3 per cent U_3O_8.

HENRY CLAY.

Although the Henry Clay claim had been prospected rather extensively before it was leased by the National Radium Institute, and promised to be a good producer, the amount of ore finally discovered was disappointing. Two prospect holes showed considerable vanadium ore, but no signs of uranium ore other than a slight yellow stain along joints. The principal pocket of carnotite was in the bottom and along the sides of a small gulley near the center of the claim. This pocket measured about 10 by 25 feet, and the ore was 4 inches to 1 foot thick. During 1915 about 10 tons of shipping ore

and 3 tons of milling ore were taken from this pocket. Mining was chiefly by open cut, although a little carnotite was found near the edge of the main pocket in a tunnel driven about 20 feet on a vanadium-bearing lead in the sandstone. A similar tunnel along a similar promising lead on the opposite side of the gulch was without result. About 2½ tons of shipping ore and 5 tons of milling ore were obtained from a small pocket discovered about 150 feet farther up the gulley in 1915. No more ore was found, although over 37 per cent of the time spent in work on this claim was prospecting. The uranium content of the ore was not high.

MEDEA.

Three pockets of ore were found on the Medea claim. The first had been opened, before the claim was leased, by an open cut about 10 feet long, 5 feet wide, and 8 feet deep, from the end of which a tunnel had been driven for perhaps 25 feet. Several tons of ore were obtained by the National Radium Institute from the sides of the cut and by stoping in the tunnel. This ore averaged fairly high in vanadium, but only a little better than 2 per cent U_3O_8. The second pocket was discovered about 75 feet east, in the same ledge of sandstone. This pocket was found at the surface, but it soon pitched downward, and was followed by a tunnel about 15 feet, when it widened, resulting in a small stope measuring 12 by 15 feet. The ore from this stope was much better than the average from the whole claim, some of it carrying 3.5 per cent U_3O_8. In this stope were found the fissures filled with calcium vanadate and vanadic oxide [a] mentioned on page 44. The third pocket was discovered about 200 feet farther east along the ridge and was also opened by tunneling. It proved to be smaller than either of the others. After this tunnel had been driven about 12 feet the ore ran so near the surface that the roof was taken down; the resulting open cut, about 5 feet wide and 7 feet deep, extended some 15 feet farther. The carnotite here was above the average in vanadium, but rather low in uranium. Its color was decidedly greenish-yellow, and the impregnated sandstone was so evenly bedded as to have almost the appearance of shale.

The Medea was so far from camp and the ore was in general so poor that only one miner was steadily employed, who worked this claim during the whole or a part of 10 of the 17 months. As little prospecting or development was necessary on the Medea, the output of ore, 900 sacks, was excellent for the amount of work done. This was the only claim of the group worked by the National Radium Institute that averaged more than 4 sacks per man per shift, though the cost of production, which included the upkeep of a saddle horse required by the miner, about equals that on the Maggie C.

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[a] Hillebrand, W. F., Merwin, H. E., Wright, F. E., Hewettite, metahewettite, and pascoite, hydrous calcium vanadates: Proc. Am. Phil. Soc., vol. 53, 1914, pp. 31-54.

BUCKEYE AND NOONTIME.

During assessment work for 1914 on the Buckeye and Noontime claims, several small pockets of ore were discovered and mined, producing in all about 8 tons of shipping ore, most of it from the Buckeye. A small quantity of milling ore was mined and stored.

Most of the ore found on the Bitter Creek claims was low grade, and because of their distance from the mill and their relative inaccessibility no attempt has been made to treat this ore, although it remains available in case the market price for concentrates should warrant hauling it to a mill. During the assessment work for 1915, only 275 pounds of ore was mined from these two claims. In 1914 the assessment work was done from a small subsidiary camp situated on Bitter Creek, the gulley previously mentioned, along which the claims are located. In 1915, however, the miners doing assessment work were taken to Bitter Creek daily from the main camp.

DIXIE.

Although the Crucible Steel Co. is said to have obtained several sacks of high-grade ore from two "bugholes" on the Dixie claim, the National Radium Institute, after weeks of diligent prospecting, was unable to find any ore at all. The most promising indication was a greenish-yellow vanadium stain on one of the sandstone ledges overlooking East Paradox Valley, but a tunnel more than 25 feet long found no ore there, and in fact no other indication of ore. Under ordinary circumstances this claim would have been promptly dropped, but as the terms of the contract required the National Radium Institute to perform the annual assessment work on all unpatented claims in the Long Park group, the prescribed amount of such work was spent in careful prospecting, as on the Uranite and the Noontime.

VANADITE.

On the Vanadite claim the chief pocket of ore found was in the rimrock of the main ridge above the camp. The pocket extended along the face of the cliff about 30 feet; it was opened by several short tunnels but proved very shallow, the ore invariably giving out at 10 or 12 feet from the face of the cliff. This ore contained more than the average vanadium content, and had a dark greenish-gray color. The uranium and the vanadium mineralization was in more or less well-defined alternate bands. Extensive prospecting revealed only one other pocket on this claim, a small lens at the west end which was taken from a 10-foot tunnel. The total production was about 8 tons of shipping ore and about 10 tons of milling ore, all produced in 1914, and 2 tons of milling ore mined in 1915. The uranium content of all ore from this claim was low.

URANITE.

On the Uranite claim, a subsequent location to both the Henry Clay and Vanadite claims, which it overlaps (See Plate II, page 14), the annual assessment work showed no ore whatever, either of milling or shipping grade, in any part of this claim not included in the other two claims.

PROSPECTING THE CLAIMS.

As the carnotite ore bodies on the Long Park claims were invariably irregular masses or pockets rather than well-defined veins or strata thorough search for such pockets was necessary. Some of the ore bodies contained only a few hundred pounds, but a few occasionally produced as much as 100 tons, although the latter instances were extremely rare. Hence, to insure a steady supply of ore required constant prospecting. Prospecting started when the National Radium Institute began work and continued throughout the period of production. In this way a fairly steady output was maintained in spite of the patchy and irregular distribution of the ore which often gave out suddenly at any one working.

Prospecting methods had to be adapted to the requirements of each claim or situation, and no one indication could be followed with the certainty of finding ore, although there are a number of signs that may indicate its presence. Perhaps the most promising of these was the discovery of float, either a piece of weathered, shaly, rather fine-grained sandstone impregnated with some dark gray-green mineral, or a piece of coarse-grained sandstone containing traces of carnotite and vanadium leached or otherwise weathered. When float was found, the customary methods of prospecting generally led to finding the mother ledge. If the float was of the first mentioned variety, this ledge was usually a black band of mineralized sandstone in which were sometimes seen minute grains of carnotite. In the second instance the grains of carnotite in the ledge were larger and more abundant, often giving the sandstone a yellow tint. In either case, however, the surface indications had to be followed by a test pit, open cut, shaft, or tunnel, according to the topography, and even with all signs favorable the presence of a paying body of ore could not be predicted with any accuracy. On one claim in particular where the signs seemed very promising, weeks of prospecting uncovered no carnotite.

DRILLING.

In prospecting claims where the ore was commonly found close to the surface, a hand churn drill was often a great help. In using this drill, which is a bar of drill steel 12 to 14 feet long and sharpened at

both ends, a vertical hole 4 or 5 feet deep is drilled with an ordinary drill; then the miner inserts the long drill and works it up and down, the drillings being removed with a scraper. With this simple tool it was possible to drive a hole 8, 10, or 12 feet even in hard sandstone. The chief disadvantage lies in the somewhat uncertain indications of the nature of the strata obtained from the sludge. Such a hole can not be kept very clean, so that the exact depth of any ore found is obscured by the particles of ore being mixed with sludge formed in drilling through the rock above. The entrance of the drill into the rock below the ore will be obscured in like manner, so that the exact thickness of the ore body can not be determined. Nevertheless, the churn drill was extremely useful, indicating advantageous places at which to sink shafts and giving approximately the probable depth to ore.

Where the number of workable claims and the amount of prospecting to be done are sufficient to warrant the extra expense, core drills should be used. There are a number of core-drilling machines on the market, some of which have been tried by one company working claims in the carnotite district, with results said to be satisfactory. The advantages of the core drill, which brings up a cylindrical section of the strata being penetrated, over the churn drill are obvious. However, the irregular and pockety nature of the carnotite ore bodies makes even core drilling uncertain unless the holes are closely spaced, for otherwise a number of holes on a claim may miss an important pocket. On the other hand, core drilling should be much less expensive than sinking a shaft, no matter how small, to an equivalent depth. Holes 30 and 40 feet deep were obtained by the company mentioned above and in the absence of surface indications shafts are the only available substitute for drill holes. One objection to using a core drill in Long Park was the amount of water required. The authors are advised that about 60 gallons a day were used with the machine mentioned above, in addition to such water as could be collected and used over again. Where water must be hauled even for drinking and cooking purposes, the use of any considerable amount for drilling becomes a serious consideration.

CHARACTER OF ROCK.

In places the character of the rock was a good indication. The ore was usually found in the more porous parts of the ore-bearing sandstone rather than in the close-grained and highly indurated parts, hence the miners did not expect ore when the rock was "tight," but its beginning to "loosen" was considered a good sign. The ore was often found near and below a sandstone which, upon weathering, had a peculiar structure, resembling a conglomerate with pebbles the size of marbles. This appearance may have been due to differences in the cement, the softer part being eroded by the weather and

leaving the harder parts, although no difference in the nature or texture of the unweathered sandstone was visible. Some ore bodies were discovered solely through the presence of this pseudo-conglomerate. Calcite was often closely associated with the ore, so that in some places its presence led to the discovery of ore. Also carbonaceous material in varying amounts was generally associated with the carnotite, and often served as an indicator.

Places suitable for prospecting were sometimes selected solely on account of the strike and dip of the strata, although the irregular character of the ore bodies frequently made this method uncertain. At some places ore showed plainly at the outcrop; at others a vanadium stain, grayish, greenish, or black, sometimes led to ore when properly followed. The nearest approach to an actual vein was discovered on the Florence claim, where an ore-bearing stratum was practically continuous for 150 to 200 feet. On the Maggie C the ore-bearing zone, though relatively large, was not continuous or along well-defined planes, as prospecting on the farther side of the Maggie C gulley failed to show any evidence of ore. The ore dips in that direction, however, and may lie too deep there to be reached except by drilling.

GENERAL PROCEDURE IN DELIMITING ORE BODIES.

In short, no rule in prospecting for carnotite can be given, except as good judgment might be shown in choosing a favorable situation, and then the actual location of an ore body required at least two or three days and often several weeks of continuous work. The nature of this discovery work depended naturally on the topography and the probable depth of the ore-bearing horizon. On the Maggie C the ground sloped gently from a ridge into a gulley, and the ore, which as a rule dipped slightly more than the surface of the hillside, was 6 inches to 12 feet below the surface. On the higher ground removing a few inches of soil in many places uncovered the ore, which was found literally at the grass roots. Farther down the slope shallow pits, often supplemented by trenches, were dug. At still greater depths shafts were sunk 8, 10, or even 12 feet, and tunnels were driven into the face of the gulley to reach the ore from below.

On the Florence and Nellie claims, where the ore was near the surface, shallow pits and trenches were dug. Sometimes where the surface soil was a foot or so in depth a heavy steel bar pointed at one end and having a hook at the other was driven down to bedrock with a sledge and then pulled up by means of the hook. A small charge of dynamite exploded in the hole loosened the surrounding soil and facilitated its removal. In the gulleys most of the prospecting was by small tunnels, about $4\frac{1}{2}$ by 6 feet in cross section. In two instances, previously mentioned, these followed "bug holes" or channels in the

rock caused perhaps by a fossil tree trunk or a root, which had become filled with high-grade carnotite. In places the sandstone surrounding the hole was impregnated with carnotite, forming a good shipping ore, to a depth of 12 or 15 inches. In other places the surrounding sandstone was barren.

The Great Western claim covered a still more rugged hillside and a gulch, and much of the prospecting was by short tunnels. This claim, however, differed from all the others in that a considerable part of the ore was found at a depth of 3 to 8 feet in the partly eroded brecciated zone of a horizontal, or nearly horizontal, fault. This condition made prospecting uncertain, as pieces of ore were mixed indiscriminately with rock, sand, and clay, and there were no reliable indications. As the miners said, prospecting was like looking for potatoes in a potato patch without rows. In prospecting this part shallow pits or trenches were dug more or less at random.

On part of the Vanadite claim the ore occurred beneath the edge of a sandstone ledge from which the ground sloped abruptly. Here prospecting was by short tunnels, some driven through the talus on the slope and some into the exposed sandstone. The rest of the claim presented no special problems of prospecting.

TIME DEVOTED TO PROSPECTING.

The first table following shows the amount of time devoted to prospecting and assessment work on each claim in percentages of the total number of 8-hour man-shifts worked each month on the entire group of claims. An average of 14 per cent of the total time per month was spent in such work, the amount falling off appreciably in midwinter. On the four largest producers the averages follow in general the total production, except that more prospecting was done on the Nellie than on the Great Western, whereas more development work was necessary on the Nellie, owing to the topography.

The second table shows what percentage of the monthly number of shifts worked at each individual claim was spent in prospecting; it shows that relatively much less prospecting was necessary on the Maggie C, where the ore was fairly plentiful and reasonably well defined as compared with the Florence, Nellie, and Great Western, the other chief producers. This table shows which claims were steady producers and which were nonproductive, that is, where the month's work was spent in prospecting, such as the Dixie and the Uranite, and those on which the ore "came hard," as on the Henry Clay, Buckeye, and Vanadite. On the Medea two or three small pockets, varying greatly in quality, afforded fairly steady work for one man, thus necessitating relatively little prospecting. The figures in the table may be taken as typical for a small group of carnotite claims in the region under consideration.

Time spent in prospecting and assessment work on each claim, in percentages of total mining shifts per month for all claims.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.	Total.
1914.											
June	1.5	....	....	0.0	0.0	....	....	....	....	....	1.5
July	3.8	1.0	2.1	1.6	1.6	....	....	....	....	....	10.1
August	1.4	2.8	2.4	2.1	.6	1.1	....	....	....	....	10.4
September	.0	.0	.0	.0	.0	.1	0.0	....	0.6	....	.7
October	1.0	1.7	.2	.9	3.7	.0	3.1	2.3	3.4	2.1	18.4
November	3.2	3.0	1.0	2.7	....	....	....	3.8	.2	3.9	17.8
December	7.0	1.9	3.7	1.6	....	.0	....	....	2.7	.3	17.2
1915.											
January	3.8	1.5	3.1	.9	....	....	....	....	....	....	9.3
February	1.5	.1	1.2	1.1	....	....	....	....	....	....	3.9
March	2.1	.0	.0	.5	....	....	....	....	....	3.1	5.7
April	3.5	1.6	.6	4.0	3.2	.3	....	2.3	3.5	2.4	21.4
May	1.8	2.9	3.2	.0	....	....	....	2.0	1.9	....	11.8
June	4.8	3.3	6.6	3.1	2.3	....	6.0	1.1	....	....	27.2
July	8.1	3.7	1.9	1.3	.0	....	....	....	....	....	15.0
August	5.0	5.1	.0	3.4	.1	.5	....	....	....	....	14.1
September	5.1	9.8	5.7	1.6	.8	.2	....	....	....	....	23.2
October	3.4	7.8	4.8	.9	....	.0	13.2	....	....	....	30.1
Average	3.6	2.7	2.1	1.5	.7	.1	1.3	.6	.7	.7	14.0

Time spent in prospecting and assessment work on each claim, in percentages of total mining shifts per month for each claim.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.
1914.										
June	2.9	....	....	0.0	0.0	....	....	....	....	....
July	6.1	46.3	27.0	12.8	26.1	....	....	....	....	....
August	2.1	34.0	44.4	26.7	35.0	43.3	....	....	....	....
September	.0	.0	.0	.0	.0	6.2	0.0	....	100.0	....
October	1.7	28.2	13.3	16.6	76.6	.0	44.0	100	77.3	100
November	5.6	48.0	66.6	24.7	....	....	....	100	2.1	100
December	10.4	63.6	37.0	17.7	....	.0	....	....	55.5	100
1915.										
January	5.3	100.0	26.3	9.6	....	.0	....	....	....	....
February	2.1	1.0	21.1	10.3	....	.0	....	....	....	....
March	3.4	.0	.0	4.4	....	.0	....	....	....	100
April	6.5	10.1	100.0	35.3	97.3	18.1	....	100	100.0	100
May	2.7	28.2	58.1	.0	....	....	....	100	100.0	....
June	1.3	14.6	84.4	17.2	100.0	....	100.0	100	....	....
July	20.6	12.2	28.1	11.9	.0	.0	....	....	....	....
August	17.1	9.7	.0	60.1	2.6	9.1	....	....	....	....
September	23.4	18.0	61.4	40.0	20.0	.0	....	....	....	....
October	18.6	31.9	18.6	6.1	....	.0	100.0	....	....	....

CONDITIONS GOVERNING DEVELOPMENT.

The 11 claims leased by the National Radium Institute may be considered a normal group, as the Maggie C, an excellent producer, was not without rivals, and three of the claims showed no ore at all, the assessment work on these being conducted at a dead loss and continued only to fulfill the terms of the lease. This condition served to balance the proportionately large tonnage mined from the Maggie C. The table on page 40 shows that the Maggie C produced

67 per cent of the total sacks of ore mined, the actual figures for tonnage being approximately 66½ per cent. Some may consider this an undue proportion from one of a group of 11 claims, and thereby designate the Maggie C as exceptional.

However, if the Maggie C had not produced as freely as it did, more extensive work would undoubtedly have been done on the other leased claims. Although this claim was a splendid producer the ore did not come easily, for the development work probably exceeded that done on any other carnotite claim in the district. None of the Maggie C ore was of very high grade, much of it barely averaging 2 per cent U_3O_8, while 56½ per cent of it was from underground work, involving planning, uncertainty, and expense. On a large part of this claim, that is, the north side of the gulley and the flat beyond the ridge, no ore was found.

However, if the Maggie C and the three unproductive claims be eliminated and the Florence, Nellie, Henry Clay, Great Western, Medea, Buckeye, and Vanadite claims be assumed to form an average group, to produce from these seven claims quantity of ore equal to that produced from the 11, during a like period by the same number of men, would have required about 20 such claims. This estimate is, of course, hypothetical because, with more men at work on these claims, more development and hence a much larger output is to be expected. As the development of the carnotite district progresses it is probable that a number of claims as productive as the Maggie C will be discovered; in fact, several claims in this district have yielded a far larger tonnage than the average. The Joe Dandy, on the south rim of East Paradox Valley, the Cliff mine at Saucer Basin, and the Last Chance, in the Hydraulic region, with their deposits of high-grade ore, are ample proof that claims yielding a large output of carnotite are not exceptional.

In general, however, the investor in carnotite mines should remember that a few scattered claims will not insure the continuous or sufficient ore supply necessary to run a radium-extraction plant or even to conduct a paying mining proposition. At least 20 claims, carefully chosen, will be necessary for a safe investment. Also the fact that a claim shows an outcrop or adjoins a known producer is not conclusive proof of its excellence, as the deposits are not continuous, nor can their depth in any instance be predicted. Careful prospecting and development work, with a rough outlining of possibilities and some idea of eventualities, should precede the purchase.

DEVELOPMENT.

Prospecting comprised searching for indications of ore and following these by open cut, churn drill, or other means. Development included following a lead of low-grade ore until it developed into a work-

able body of ore, delimiting the ore body to determine its possibilities, or, sometimes, driving a tunnel or open cut toward a known body of ore otherwise inaccessible.

When the ore lies at or near the surface, as on two or three of the Long Park claims, it can be easily and cheaply opened by means of pits, shallow shafts, or open cuts. The working place being well lighted, it is possible to drill and blast more effectively by taking advantage of changes of direction and quality, and the preliminary sorting of ore from waste can be made with greater accuracy. It is almost impossible to sort carnotite ore by artificial light, for reasons which are explained later. Where the amount of cover was too great to permit economical removal, mining had to be underground, and although excavation was somewhat more difficult, the protection of the miner from the weather and the fewer interruptions to work offered some advantages. During midwinter in Colorado and Utah the frozen ground makes surface mining impracticable, and one of the objects of development was to open in the fall suitable underground places for winter work.

With surface mining there is less need of development, because the miner can follow and delimit the ore more easily and handling waste rock is simplified. However, some development work is necessary to determine the disposition of waste (in order to avoid covering up possible ore bodies) and to minimize the handling of waste and ore. To this end, in open-cut work, development cuts wide enough for a wheelbarrow, or on the Maggie C a mine car and track, were driven along one side of the ore body. Waste could then be piled against the barren side of the cut and the cut widened until the other side of the ore body was reached. This method saved rehandling waste or tramming it an unnecessary distance. In mining a large ore body, as on the Maggie C, auxiliary open cuts were dug at intervals sufficient to provide working places, and a fan-shaped track system was laid. With the ore body thus defined, the handling of waste was simplified and to some degree the irregular character of the ore deposit overcome. All work driven along the side of the deposit, whether in a trench or underground, was called development unless such work produced a normal amount of shipping ore—that is, approximately 200 pounds per man per shift.

Underground development work was required in order to limit the ore body, to provide more working places, and also to provide for the advantageous handling of waste. Another and even more important factor in underground work is the greater cost of mining in narrow work while driving the tunnels as compared with that of stoping ore blocked out by tunnels on two or more sides. The cheapness of the latter method well repaid driving a system of devel-

opment tunnels and crosscuts wherever the ore body proved to be of noteworthy size or importance.

One feature of the development work which was of especial advantage on the Maggie C, and was used on the other claims where conditions were favorable was driving raises from various points underground to the surface to insure good light and air. This method, where the distance to the surface is not more than 8 to 12 feet, has many advantages. The better air enables the men to work to the best advantage, the time required for the smoke to clear after blasting is reduced to the minimum, and most important of all, the preliminary sorting of the ore can be done accurately and quickly and at the working place. Experience has demonstrated that time spent in such development work was repaid many times.

OPEN-CUT AND TUNNEL WORK COMPARED.

Choice must frequently be made between open-cut and drift in opening or developing a lead of low-grade carnotite ore. It is, of course, impossible to establish arbitrarily a definite point at which the open cut is preferable, but experience has shown that the drifting or tunneling is usually preferable wherever the overburden is more than 5 feet thick. At times, however, a tunnel is advisable when the overburden is thinner, as, for instance, when the waste must be carried far or the overburden forms naturally or through freezing a satisfactory roof. Obviously, expensive work such as tunneling should not be undertaken without more than an even chance of its developing enough ore to pay expenses at least. Hence in the course of the work described in this chapter, development tunnels were never started except to drive, usually through barren rock, to a body of ore that had been located by prospecting from shafts or pits, or to follow "bug holes."

The following table shows the cost of driving a tunnel on the Maggie C claim through barren sandstone to reach an ore body discovered by a test pit. This tunnel, which was 49 feet long, averaged 4 feet 2 inches wide and 6 feet 2 inches high. Two men were employed during the day and one man part of the time at night. The waste was wheeled in barrows 15 to 40 feet from the entrance. A slip or joint-plane in the rock, which was followed along one side of the tunnel for the entire length, was of much advantage in blasting. A comparison of the figures for driving through barren sandstone with the figures in the table at the bottom of page 26 shows the lesser cost driving in ore.

Cost, exclusive of overhead charges, of driving development tunnel in barren sandstone on Maggie C claim.

[Tunnel 49 feet long, 4 feet 2 inches wide, and 6 feet 2 inches high.]

Item.	Cost per linear foot.	Cost per cubic foot.	Total cost.
Foreman	$0.51	$0.018	$25.00
Mining	2.94	.105	144.00
Mucking	1.44	.052	70.50
Blacksmithing	.47	.017	23.19
Candles	.05	.002	2.55
Powder	.79	.028	38.48
Fuse	.09	.003	4.47
Caps	.07	.003	3.57
Total	6.36	.228	311.76

Instances of tunnels driven along "bug holes" are the main tunnel on the Florence claim, where the small amount of ore removed just about paid expenses until the larger pocket was encountered, or those on the Maggie C claim, driven through milling ore too low grade for sacking and shipment, but giving promise of leading to a pocket of shipping ore. The table following shows the cost of driving such a tunnel 78½ feet, average width 5 feet 9 inches, mean height 5 feet 10½ inches. One miner did the drilling and blasting, although part of the time another man was employed shoveling up the débris, and wheeling it in a barrow to the mouth of the tunnel and tramming it by mine car about 175 or 200 feet to the dump. The tunnel was driven for the most part through milling ore, but near its end a pocket of shipping ore was discovered and mined.

Cost of driving mining tunnel on Maggie C claim.

[Tunnel 78½ feet long, 5 feet 9 inches wide, 5 feet 10½ inches high.]

Item.	Cost per linear foot.	Cost per cubic foot.	Total cost.
Foreman	$0.199	$0.0059	$15.62
Mining	2.427	.0718	190.50
Mucking	1.146	.0339	90.00
Blacksmithing	.223	.0066	17.50
Candles	.027	.0008	2.15
Powder	.266	.0079	20.90
Fuse	.042	.0012	3.30
Caps	.035	.0010	2.75
Total	4.365	.129	342.72

From four of the five best producing deposits, two on the Maggie C and one each on the Florence and Great Western, most of the ore was mined by tunnel and stope, and approximately 60 per cent of the total production was from underground work.

TIME SPENT IN DEVELOPMENT WORK.

TOTAL TIME.

The writers found that about 12 per cent of the total time spent on all mining operations was profitably employed in development. This is an average for all claims, the figures varying from a negligible quantity on claims where no ore was discovered to 8.4 per cent on the Maggie C, where the development work was most thorough and systematic. To this must be added the time spent in mining milling ore and charged to the mill account. This work sometimes served as development by discovering or opening up pockets of shipping ore. Comparison of the following table with the table of ore produced by each claim (see p. 40) shows a relation between development and production that deserves serious consideration.

Percentages of entire mining time spent in development work.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.	Total.
1914.											
June........	3.8	....	....	0.0	0.0	....	....	....	....	....	3.8
July........	12.8	0.4	0.8	2.5	.5	....	....	....	....	....	17.0
August........	5.0	1.6	.9	3.3	.2	0.0	....	....	....	....	11.0
September....	.0	.0	.0	.0	.0	.0	0.0	....	0.0	....	.0
October......	11.1	2.9	1.4	2.1	.8	.2	.3	0.0	.1	0.0	18.9
November....	12.4	1.8	.4	1.9	....	....	....	.0	1.3	.0	17.8
December....	11.2	....	.5	2.5	....	.0	....	....	1.6	.0	15.8
1915.											
January......	13.5	.0	1.7	4.1	....	.2	....	....	....	....	19.5
February.....	8.6	.0	.4	3.6	....	.8	....	....	....	....	13.4
March........	7.1	.0	.0	3.9	....	.6	....	....	....	.0	11.6
April........	13.0	.6	.0	1.4	.0	.0	....	.0	.0	.0	15.0
May..........	17.6	.4	.5	3.6	.0	....	....	.0	.0	....	22.1
June..........	5.5	.9	.0	.6	.0	....	.0	.0	....	....	7.0
July..........	3.2	.6	.0	.9	.4	.2	....	....	....	....	5.3
August........	3.2	1.3	.0	.0	.3	.2	....	....	....	....	5.0
September....	2.9	5.1	.4	.0	.2	.2	....	....	....	....	8.8
October......	3.5	1.3	2.2	.4	....	.0	.0	....	....	....	7.4
Average.	8.4	1.0	.5	1.9	.15	.14	.01	....	.2	....	12.0

TIME SPENT ON EACH OF THE VARIOUS CLAIMS.

The following table shows the percentage of the actual working time spent each month in development work on the different claims. On the Great Western, laid on a hillside, the ore bodies were usually small and irregular and relatively more time was spent in development than on the Nellie. The relatively extensive development on the Vanadite uncovered chiefly milling ore. On the Nellie low-grade ore was widely distributed, but no large body of shipping ore was uncovered. The Maggie C (see Pl. II, p. 14) was the most extensively developed, being opened on three levels, by open cuts, stope, and tunnel.

Percentages of total time spent in development work.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.
1914.										
June	7.3	....	....	0.0	0.0	....	....	....	....	....
July	20.4	19.5	10.4	19.5	8.3	....	....	....	....	....
August	7.0	19.6	16.9	42.3	10.0	0.0	....	....	....	....
September	.0	.0	.0	.0	.0	.0	0.0	....	0.0	....
October	17.7	46.9	71.1	37.9	1.7	25.0	3.6	0.0	18.9	0.0
November	21.2	26.6	22.2	17.7	....	....	....	.0	15.6	.0
December	16.4	.0	4.6	27.0	....	.0	....	....	13.3	.0
1915.										
January	18.7	.0	14.1	42.6	....	12.0	....	....	....	....
February	12.2	.0	6.5	33.5	....	16.6	....	....	....	....
March	11.3	.0	.0	34.3	....	12.5	....	....	....	.0
April	24.2	3.9	.0	12.9	.0	.0	....	.0	.0	.0
May	26.5	3.6	8.5	43.2	....	....	....	.0	.0	....
June	14.8	3.9	.0	3.7	.0	....	.0	.0	....	....
July	8.0	1.9	.0	7.7	10.0	4.8	....	....	....	....
August	11.3	2.4	.0	.0	10.2	3.0	....	....	....	....
September	13.6	9.4	4.6	.0	20.0	2.7	....	....	....	....
October	19.2	5.4	8.4	.3	....	.0	.0	....	....	....

REMOVING ORE.

All work incidental to the mining of shipping ore, its sorting, and preparation for shipment is included under "removing ore." In carnotite mining the waste rock and the ore are always removed separately, to prevent the mixing of ore and waste, and to obviate the shattering of the ore, and consequent loss of carnotite, by the heavier charges used in blasting rock. For this reason the two operations constitute separate, though related, problems. On the Maggie C claim, because of the number of miners employed and the quantity of ore produced, the workmen could specialize on definite types of work and the data could be kept in more detail. The following table shows the proportion of time required at this claim for the various mining operations included under "removing ore," such as mining and sorting the ore, and filling, sewing, and piling sacks.

Detailed data for Maggie C claim showing percentage of entire mining time spent in various operations.

	Per cent of total time.
Prospecting	7.6
Development	14.7
Stripping	19.8
Mining	24.0
Mucking	17.6
Sorting	7.9
Sewing sacks	2.6
Filling sacks	.7
Laying track	.8
Packing and loading	.5
Piling sacks	.3
Picking dump	2.8
Miscellaneous	.6

The work in mining carnotite falls naturally into two classifications: Surface work and underground work. Surface mining at the claims of the National Radium Institute consisted usually of gradually enlarging, as the ore body was followed, the pits or trenches; the underground work consisted of tunneling and stoping. In both classes of work the drilling and blasting methods were much the same, although of course more simple in surface work than underground. Also, as the preliminary sorting of waste rock from ore was much more difficult underground, it was almost always necessary to bring the ore to daylight before sorting it.

DRILLING AND BLASTING.

In all carnotite mining hand drilling is employed exclusively. On the claims of the National Radium Institute each miner drilled and blasted his own holes. The single-jack or striking hammer used weighed about 3½ pounds and had two striking faces; the handle was about 14 inches long.

The drills were made of octagonal steel. At first, ⅞-inch steel was purchased. It gave excellent results, but subsequent tests showed that 13/16-inch steel was just as durable and efficient, and had the advantages of lower first cost and easier transportation from the blacksmith shop on the Maggie C to the other claims. Both sizes of steel were sharpened to a chisel edge. Usually the steel was cut into four lengths: Starters 18 inches, seconds 24 inches, thirds 30 inches, and fourths 36 inches, although for deeper holes drills 48 and 60 inches long were sometimes made. The gage of cutting edge for each length was the same for both weights of steel, that of the starters being 1¼ inches, seconds 1⅛ inches, and thirds and fourths 1 inch.

Drill holes were ordinarily 2 to 2½ feet deep. The sandstone varied in hardness, and the rate of drilling the waste rock was governed largely by its hardness and the presence or absence of black vanadium ore. Milling or low-grade ore was usually softer than the waste and the shipping ore was invariably softer than either. The average speed of drilling in waste rock was 37 inches an hour, in milling ore 42 inches an hour, and in shipping ore the rate was slightly more. In surface work the rate of drilling naturally averaged slightly higher than in underground.

The placing and direction of the holes were governed by the conditions at the working face, advantage being taken of any "slips" or joint planes or of any benches or other free surface left from a previous blast. Even in the tunnel work any fixed arrangement or round of holes was not practicable, though usually possible in other mining. Carnotite ore is so friable that the minimum shock that will loosen the ore is desirable, in order to prevent finely broken ore

being lost in the débris, and the hand drilling permits much greater elasticity of arrangement than the use of machine drills. On the average, two holes per shift per man were sufficient to shatter and loosen the shipping ore so that it could be broken down and removed. In the underground work blasting was usually done either at noon or at the end of the day, in order to permit the smoke to clear with the least possible loss of time by the miner. In blasting ore, after the drill holes were finished and before they were loaded, the floor for some distance from the face was carefully cleaned of waste, and was then either swept with a broom, or a sheet of canvas was laid on it to catch the fine soft material which was most valuable and most easily lost.

EXPLOSIVE AND FUSE USED.

All blasting was done with gelatin dynamite of 40 per cent strength. No experiments were made with dynamite of other grades because of poor results obtained with both 60 per cent and 30 per cent strengths elsewhere. The average charge in mining shipping ore varied from three-fourths to one stick per hole, according to the quality of the ore; in blasting waste the charge was one and three-fourths to two sticks per hole; in prospecting and development work the average was sometimes higher. White-finished gutta percha fuse was used. Some difficulty was experienced during the winter months in persuading the miners that cold fuse should not be handled roughly and should be warmed before being uncoiled for use. The amount used per hole varied, but the minimum length permitted was 2 feet, as nearly as the miner could estimate. No. 6 detonators were found much superior to No. 5 detonators in lessening the number of missed holes and increasing the breaking power of an equal charge of dynamite.

SORTING AND SACKING.

After blasting, and, if underground, close inspection of the roof, the scattered pieces of shipping ore were carefully collected by the miner and placed in a pile, any stray pieces of waste were thrown into another pile, and the low-grade ore or milling ore into another. The miner then removed with pick and gad all ore and rock loosened by the blast. This material received a preliminary sorting. In surface work, and in underground places sufficiently near a raise for good illumination, three products were made: Shipping ore, milling ore, and waste. Where the light was poor the material was separated into only two grades, ore and waste, the former being taken to the sorting table and there separated into shipping and milling ore. On claims where the number of men employed made it practicable, the shipping ore from the preliminary sorting was taken in

empty powder boxes, wheelbarrows, or mine cars to the sorting table, where another workman made the final sorting for shipment. On claims where only one or two men were employed the miners who, of course, had sufficient experience sorted the ore for shipment and put it in the sacks, which they sewed at convenient times

ILLUMINATION.

Various experiments were tried to discover some means by which carnotite ore could be distinguished underground. The usual method of illumination in the workings was the ordinary miner's candle. By candlelight the yellow color of the carnotite could be differentiated only with great difficulty from the white or gray parts of the barren rock, as well as from the dark vanadium ore with which it usually occurred, making it practically impossible to do even preliminary sorting. Acetylene lamps gave better illumination, but did not make sorting appreciably easier, nor did a hand electric flash lamp with a tungsten filament nor an incandescent mantle gasolene lamp. Therefore the use of candles was continued, and where possible the ore was opened in such a way as to permit all sorting to be done by daylight. Where daylight could not possibly be obtained near the working face, by driving a raise or otherwise, the ore had to be carried to it.

TIMBERING.

Little timbering was required in the tunnels, owing partly to their being narrow (usually not more than 4½ or 5 feet) and partly to the roof being solid. In the stopes the roof was supported with pinon posts cut from growths near the workings by the miners as needed, or cut by a man detailed for this purpose and hauled to the claim by team. As the timbers were unseasoned and the bark was not removed, they were liable to decay more rapidly, but owing to the short time they were needed this consideration was not serious. Most of the worked-out stopes were promptly filled with waste rock. The posts used averaged about 8 or 10 inches at the butt and were spaced more or less irregularly as required, but the average in most of the stopes was 4 or 5 feet apart. A view in one of the stopes is shown in Plate IV, *A*.

TIME SPENT IN REMOVING ORE.

TOTAL TIME.

The following table shows what percentage of the total monthly shifts, for the entire group of claims, was spent in removing ore. Approximately one-third of the total work was spent in such operations, and 60 per cent of this time was employed on the Maggie C

claim. By comparing this table with the one on page 40, showing the production from the various claims, the analogy will be seen to be striking. In other words, the presence and the nature of the ore body having been established by prospecting and development, the mining of the ore required a certain proportion of the total time worked, which parallels closely the tonnage produced, claim for claim, except on those claims where, owing to the irregular character of the ore deposit, the results of mining operations were necessarily uncertain.

Time spent in removing ore on each claim, percentage of total mining shifts for the group of claims.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.	Total.
1914.											
June	36.4	......	......	10.7	14.5	......	......	......	......	......	61.6
July	26.9	0.8	3.8	7.8	2.6	......	......	......	......	......	41.9
August	37.3	2.6	1.4	1.6	.7	1.1	......	......	......	......	44.7
September	33.4	1.7	1.2	5.5	1.7	1.1	3.6	......	0.0	......	48.3
October	24.0	1.0	.1	1.3	.2	.3	2.6	0.0	.5	0.0	30.1
November	18.8	1.5	.1	3.0	......	......	......	.0	3.2	.0	25.6
December	22.2	.5	2.7	2.5	......	.0	......	......	.2	.0	28.2
1915.											
January	20.2	.0	3.6	2.2	......	.7	......	......	......	......	26.7
February	29.3	2.0	1.6	2.9	......	1.8	......	......	......	......	37.6
March	23.6	5.3	.5	3.4	......	2.5	......	......	......	.0	35.3
April	13.1	5.7	.0	2.8	.1	.7	......	.0	.0	.0	22.4
May	17.4	3.1	.6	3.1	......	......	......	.0	.0	......	24.2
June	14.4	8.6	.4	6.9	.0	......	.0	.0	......	......	30.3
July	10.7	10.3	2.0	4.2	1.0	1.7	......	......	......	......	29.9
August	7.4	15.0	.1	.9	.7	2.2	......	......	......	......	26.3
September	4.4	12.7	2.3	.3	.0	3.3	......	......	......	......	23.1
October	3.7	6.7	9.3	6.7	......	.4	.0	......	......	......	26.8
Average	20.2	4.5	1.7	3.8	1.2	.9	.4	.0	.2	.0	33.0

PRODUCTION.

The tonnage mined during each month of the first year and a half of the work, 1914–15, and the number of shifts spent in order to obtain such tonnage, are shown in figure 2.

The average number of sacks of ore produced per man per shift from each claim was as follows:

Average number of sacks of ore produced per shift by each man employed.

Maggie C	3.70
Florence	2.78
Nellie	2.30
Great Western	2.76
Henry Clay	1.68
Medea	4.07
Buckeye and Noontime	1.59
Vanadite	1.72
Dixie	.00
Uranite	.00
Average sacks per shift	3.21

A. STOPE IN A CARNOTITE MINE, SHOWING TIMBERING AND FILLING WITH WASTE ROCK.

B. SACKED ORE READY FOR LOADING AND SHIPMENT

TIME SPENT ON EACH CLAIM.

The percentage of the work done each month on each individual claim that was spent in removing ore is given in the table following. This table shows which claims were not producers and explains the relation one to another of the development of the several claims. A comparison of this table with those giving similar figures for prospecting and development (see pages 22 and 27) shows the progress of the work.

FIGURE 2.—Curves showing tonnage mined at the claims during 1914 and 1915, and number of shifts spent in mining.

Time spent in removing ore, per cent of total monthly time per claim.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.
1914.										
June	68.8			93.3	95.0					
July	42.8	34.1	47.2	57.6	37.0					
August	52.3	31.4	25.4	20.9	40.0	43.4				
September	47.0	37.9	32.0	72.4	46.6	42.1	65.4		0.0	
October	38.0	17.2	8.8	24.2	5.9	50.0	36.3	0.0	11.3	0.0
November	32.4	22.6	5.5	27.6				.0	36.4	.0
December	32.7	18.3	26.9	27.0		.0			37.0	.0
1915.										
January	28.9	.0	25.9	23.5		36.0				
February	41.4	40.0	27.6	26.2		39.5				
March	37.4	48.6	41.6	29.8		48.2				.0
April	24.2	35.5	.0	25.3	.3	36.3		.0	.0	.0
May	25.9	30.0	11.1	35.7				.0	.0	
June	38.5	38.3	5.6	38.1	.0		.0	.0		
July	27.0	33.7	30.8	36.7	25.0	41.0				
August	25.9	28.6	24.0	16.5	23.0	38.6				
September	20.7	24.8	25.0	12.5	.0	42.7				
October	23.2	27.4	36.3	47.3		27.8	.0			

DETAILED DATA ON MAGGIE C CLAIM.

Detailed figures on the various phases of the work of removing ore on the Maggie C are given in the table following. Under "mining" is included drilling and blasting in ore, and the preliminary sorting in the mine. "Sorting" consisted in breaking the larger lumps of ore brought to the sorting table by the miner and discarding any material of lower grade than 2 per cent U_3O_8. Sometimes the sorter filled the ore sacks, and sometimes another man did this work and also sewed the sacks and piled them near by. Where possible, the sacks were piled near the road (see Pl. IV, *B*), the object in piling being convenience in loading and to keep the sacks off the ground to prevent their rotting, the lowest row being laid on pieces of timber, as the ore absorbs moisture rapidly.

Proportions of time spent on Maggie C in various operations included under removal of ore.

Month.	Percentage of time spent in—					
	Mining.	Sorting.	Sewing.	Filling.	Piling.	Picking dump.
1914.						
June	15.2	13.0	4.4	1.4	(a)	39.1
July	25.7	10.0	3.0	(b)	(a)	7.1
August	43.0	9.2	2.3	(b)	(a)	....
September	40.1	6.9	2.6	(b)	(a)	....
October	28.7	7.7	2.8	1.7	1.2	....
November	23.0	8.2	2.8	.9	.9	....
December	22.9	8.1	2.7	1.6	(a)	....
1915.						
January	19.0	7.4	1.8	1.7	(a)	....
February	31.2	8.3	2.1	1.9	(a)	....
March	27.3	8.1	3.1	2.0	(a)	....
April	15.1	8.0	3.0	1.1	(a)	....
May	19.7	6.3	2.4	(b)	(a)	....
June	30.7	7.8	2.6	(b)	(a)	....
July	20.7	6.4	2.0	(b)	1.0	....
August	19.3	6.7	2.1	(b)	(a)	....
September	12.2	6.1	1.7	.2	.5	....
October	13.8	6.0	1.8	(b)	1.2	....
	24.0	7.9	2.6	.7	.3	2.8

[a] Included in sewing. [b] Included in sorting.

RESORTING OLD DUMPS.

During the first two months of operation at Long Park, a considerable part of the mining time was spent in resorting and picking over old dumps. In previous work the ore had been sorted to 3 per cent grade, leaving enough ore of 2 per cent grade to warrant resorting before covering with mill ore or waste.

REMOVING WASTE.

Removing waste included drilling and blasting waste rock and low-grade ore and removing them from the mine.

DRILLING AND BLASTING.

The drilling and blasting operations were, of course, practically the same as those employed in mining the ore, except that sometimes heavier charges of dynamite were used. Care had to be taken, however, to avoid mixing the waste with the ore or shattering the latter any more than necessary, for this caused losses of the best grade of carnotite, as has been previously mentioned. In surface work the soil, rarely more than a foot thick, was carefully removed before holes were drilled and blasted. Wherever possible advantage was taken of the bench formed by the previous blast. Underground the holes were pointed to break only the waste rock from above the ore. Both on the surface and underground the rock blasted was shoveled away from the immediate face. If the number of men working on a claim, as on the Maggie C, made such a course possible, a man was detailed to remove the waste to the dump, but on claims where only one or two men were employed the mucking was done by the miner before be began to remove the ore.

DISPOSAL OF WASTE.

The disposal of waste had to be varied on each claim to meet local conditions. In many places, where the ore was in small pockets at or near the surface, the waste was shoveled to one side, after it had been ascertained that no ore lay beneath. In mining the larger deposits the usual practice was to shovel the waste back into the excavation as mining proceeded. On the Maggie C a deposit of low-grade milling ore, of considerable extent, lying 4 to 6 feet below the shipping ore, could not profitably be mined with the shipping ore but was of sufficient importance to reserve for later use in the mill. Therefore from this area the waste was transported to a dump at one side, under which the ore lay too deep to be mined by open cut, by means of a fan-shaped track system leading to the various working places. The rock dump was gradually extended and built up level with the top of the ore bin of the mill, thus disposing of the waste and providing a grade for raising the milling ore into the bin when mined later.

In the underground work on this claim the waste was handled where possible by mine cars, both in driving the development tunnels and in mining. Wheelbarrows were used for the first 50 feet of the main tunnel and in one or two places in branch headings where the end of the track, owing to the pitch of the ore, could not be extended to the face of the heading. In stoping, as much as possible of the waste was immediately filled back into the stope to support the roof if the timbers failed and to save tramming this waste outside. On the other claims the waste was removed with wheelbarrows only,

with a plank runway where necessary, but was filled back into the stopes as much as possible.

The mine cars used on the Maggie C were of 13 cubic feet capacity, two being of the scoop type and two of the standard type with a swinging door at the end. Both types were provided with a turntable and could be emptied at either side of the track. For dumping from a trestle or into an ore bin, the scoop type was preferred, but for use on a track running along the edge of a dump the standard type gave better satisfaction. Ordinary 8-pound mine rails were used, set 18 inches apart on ties spaced about a yard apart, which were cut locally from small pinon. Long-handled, round-pointed No. 2 mining shovels were used to remove the waste, but in underground work the men usually cut 10 to 14 inches off the handle for convenience in narrow quarters. The wheelbarrows were of steel, with a rated capacity of 3 cubic feet.

TIME SPENT IN REMOVING WASTE.

The following table shows the percentages of the total time spent in removing waste:

Time spent in removing waste on each claim, percentage of total monthly shifts for the entire group.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.	Total.
1914.											
June	8.5	......	......	0.0	0.8	......	......	......	......	......	9.3
July	16.9	0.0	1.1	1.3	1.1	0.0	......	......	......	......	20.4
August	25.8	1.0	.3	.6	.3	.3	......	......	......	......	28.3
September	35.1	2.6	2.4	1.6	1.9	1.3	1.6	......	0.0	......	46.5
October	24.2	.4	.1	1.6	.2	1.3	.6	0.0	.3	0.0	28.7
November	21.0	.3	.1	3.2	......	......	......	......	3.4	......	28.0
December	23.7	.5	3.0	2.2	......	.0	......	......	.2	.0	29.6
1915.											
January	30.7	.0	3.6	2.2	......	.9	......	......	......	......	37.4
February	27.7	2.6	2.3	2.9	......	1.8	......	......	......	......	37.3
March	27.4	5.0	.6	3.6	......	1.9	......	......	......	.0	38.5
April	22.6	6.9	.0	2.5	.0	.8	......	.0	.0	.0	32.8
May	28.4	3.5	1.2	1.8	......	......	......	.0	.0	......	34.9
June	15.9	9.2	.7	6.1	.0	......	.0	.0	......	......	31.9
July	15.9	12.8	2.4	3.8	2.4	2.1	......	......	......	......	39.4
August	12.2	29.5	.4	1.1	2.1	2.5	......	......	......	......	40.8
September	8.6	24.7	.7	.3	.0	3.9	......	......	......	......	38.2
October	7.0	8.5	9.3	5.8	......	1.3	.0	......	......	......	31.9
Average	20.7	6.3	1.7	2.4	.5	1.1	.1	......	.2	......	33.0

The percentages of the total monthly work on each claim which were spent in removing waste were as follows:

Time spent in removing waste, percentage of monthly total per claim.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.
1914.										
June	15.3	....	....	0.0	5.0	....	....	....	....	....
July	27.7	0.0	12.5	10.1	18.5	....	....	....	....	....
August	36.9	11.9	5.3	8.1	15.0	13.3	....	....	....	....
September	59.9	58.3	64.1	20.5	51.1	50.0	28.6	....	0.0	....
October	48.4	70.3	4.4	15.1	3.3	25.0	8.9	0.0	7.5	0.0
November	35.6	4.0	5.5	28.8	....	....	....	0.0	39.6	.0
December	34.8	15.1	30.1	23.6	....	.0	....	....	3.7	.0
1915.										
January	42.5	.0	29.8	23.1	....	48.0	....	....	....	....
February	38.4	53.3	39.8	26.2	....	40.6	....	....	....	....
March	43.3	46.0	58.3	31.5	....	37.5	....	....	....	.0
April	42.0	43.3	.0	22.9	.0	40.9	....	.0	.0	.0
May	42.5	33.6	21.4	20.5	....	....	....	.0	.0	....
June	42.9	40.7	9.3	33.9	.0	....	.0	.0	....	....
July	40.1	40.9	36.0	32.9	60.0	50.9	....	....	....	....
August	42.4	55.9	72.1	19.5	61.5	43.9	....	....	....	....
September	39.9	45.4	7.9	12.5	.0	50.0	....	....	....	....
October	36.7	34.5	36.3	41.2	....	66.6	.0	....	....	....

The following table shows in detail the percentages of the monthly time spent on the Maggie C claim in various operations included in removing waste, such as stripping, mucking, and laying track:

Time spent on Maggie C claim in removing waste.

Month.	Percentage of time spent in—		
	Stripping.	Mucking.	Laying track.
1914.			
June	7.3	5.8	2.2
July	24.3	3.4	....
August	30.0	5.6	1.3
September	25.1	23.0	1.8
October	12.5	24.4	1.5
November	10.4	24.6	.6
December	16.2	17.9	.7
1915.			
January	20.1	20.3	2.1
February	24.1	13.2	1.1
March	30.8	11.7	.8
April	23.0	18.8	.2
May	15.7	25.8	1.0
June	13.4	29.4	.1
July	19.0	21.1	....
August	20.3	22.1	....
September	24.6	15.3	....
October	19.9	16.8	....
Average	19.8	17.6	.8

cast-iron or steel plate, partly-worn jaw plates from the crusher serving for this purpose. For breaking the larger lumps or frozen ore the sorter used a 2-pound hammer provided with a square face and a chisel point. For the finer lumps he used an ordinary prospector's or geologist's pick, square-faced on one end and pointed on the other.

AMOUNT SORTED PER MAN.

The amount sorted in a day varied greatly. When the ore was of good quality the sorter had little else to do than break it to proper size for shipment and throw out any occasional piece of waste rock or low-grade ore. Under such conditions it was not unusual for one man to sort 75 to 80 sacks in one day. But when the ore was of poorer quality, and especially if shipping and low-grade ore were badly mixed, sorting was more laborious, requiring much closer attention and the handling of much more low-grade material. Under such conditions 25 to 30 sacks per sorting shift was a good day's work. The average number of sacks sorted on the Maggie C claim during the entire period of mining operations was 48.4 per 8-hour shift.

The following table gives the monthly production of sorted ore, that is, shipping ore, produced from the various claims:

Monthly production of shipping ore from the various claims, in sacks of approximately 70 pounds.

Month.	Sacks of ore produced on—										Total.
	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.	
1914.											
June	459	1	....	151	31	....	....	....	....	......	642
July	846	....	158	253	100	....	....	....	....	......	1,357
August	2,040	87	127	76	49	59	....	....	....	......	2,438
September	1,516	63	77	219	64	58	153	....	....	......	2,150
October	1,587	97	11	38	19	10	54	....	72	......	1,888
November	1,487	78	6	98	....	....	....	....	155	......	1,824
December	1,534	13	162	77	....	....	....	....	20	......	1,806
1915.											
January	1,462	11	134	116	....	45	....	....	....	......	2,068
February	1,470	52	62	107	....	92	....	....	....	......	1,783
March	1,559	315	6	106	....	118	....	....	....	......	2,104
April	1,046	359	11	118	12	42	18	....	....	......	1,606
May	1,066	228	34	113	....	....	....	....	....	......	1,441
June	644	513	33	460	1	....	4	....	....	......	1,655
July	668	440	84	267	30	76	....	....	....	......	1,565
August	466	898	4	45	24	175	....	....	....	......	1,612
September	202	544	104	21	....	217	....	....	....	......	1,088
October (1-15)	128	84	177	111	....	8	....	....	....	......	508
Total	18,480	3,783	1,190	2,376	330	900	229	....	247	......	27,535

A. SORTING CARNOTITE ORE ON FLAT ROCK.

B. CONCENTRATING MILL AT LONG PARK, COLO., SHOWING SITE

C GRIZZLY ABOVE ORE BIN OF CONCENTRATING MILL, WHERE MILLING ORE WAS RESORTED.

FILLING, SEWING, AND PILING THE SACKS.

The sorted ore was pushed to one side and at intervals shoveled into sacks 18 by 24 inches in size and holding approximately 70 pounds of ore. Burlap sacks were tried at first, but were quickly discarded in favor of double-stitched canvas sacks. These resisted wear and tear well and saved fine material, which is usually high grade, that would have been lost through the wide mesh of the burlap. The canvas sacks, of course, cost more, but the fact that some of them could be used twice, and the saving of fine material, made them much more economical than burlap in the long run.

Where the output warranted the expense, the sorter was provided with an assistant, who filled the sacks, sewed them, and piled them. To facilitate filling the sacks, a rough funnel was made of a piece of fairly heavy galvanized pipe 8 inches in diameter and 2 feet long, with a flaring top forming an angle of about 30° with the pipe and enlarging the diameter to about 18 inches. The rim of this flaring part was turned under around a circle of ¼-inch iron rod to give rigidity and better wear. This filler held the sack open and kept it upright so that the ore could be shoveled in readily.

The sacks were sewed with a 5½-inch curved needle and 16-fold cotton sail twine. The top of the sack was folded over twice, the twine being looped around one corner to form an "ear," and then sewed over and over, including the folds, until the other side was reached; a couple of half stitches formed the other "ear." The sack was then resewed in the reverse direction, two half stitches were taken around the first ear, and the twine pulled through the sack once more and the ends tied. An experienced man could sew 200 sacks in a day, the average for all claims being about 20 sacks an hour.

SAMPLING.

Before sewing a sack, a rough grab of ore, about a handful, was taken at random from it. When the sample from about 100 sacks had accumulated, it was broken by hand to approximately ⅜-inch size and carefully quartered twice; the reject went with the shipping ore and the sample was brought to camp. It was then reduced on a cast-iron bucking board to perhaps ⅛-inch pieces and quartered down to about a pound sample. This was ground on the bucking board and in a quart mortar to pass through a 30-mesh sieve, and then quartered twice, the resultant sample being tested in the electroscope.

WEIGHTS OF SACKS OF ORE.

The following table shows the average weights, each month, of the sacks of ore as they arrived in Placerville, where they were weighed, in the wagons, on a platform scale.

Average gross weight, in pounds, per sack of ore as weighed in Placerville, Colo.

Ore left Long Park in—	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.
1914.										
August	74.7			74.8	76.2					
September	72.9		75.7	77.8		62.5				
October	72.0	76.5	73.0	78.9	68.2		71.2			
November	72.0	74.6		69.0						
December	76.6		67.6		67.5	68.3			67.0	
1915.										
January	69.0		66.8	68.4						
February	68.3	66.6		66.3		64.6				
March	73.1	78.7				64.8				
April	70.6	74.4	74.0	80.0		67.7	75.0			
May	67.7	71.7	68.2	72.7		68.5				
June	68.0	72.6	70.0	84.4			67.5			
July	66.6	69.8		70.6						
August	68.6	74.6				70.4				
September	75.9	77.4	70.7	70.3	67.1	73.4				
Average	71.2	73.5	71.2	74.6	71.4	68.5	71.4		67.0	

Average gross weight per sack, 71.67 pounds.

The wide variations in these figures are doubtless due to many causes, such as the personal equation of the sorter, for although the sacks were of the same size some slight diversity in filling was possible. It was thought that the ore per sack would weigh heavier during the winter months, on the theory that the frozen ore contained more moisture. The difference in weight between a sack of ore containing 2 per cent moisture, as in summer, and one containing 7 per cent, as in winter, was not sufficient, however, from the record in this table, to increase the average appreciably during the winter months. The character of the ore on the different claims varied greatly. Some types of ore broke into fairly large pieces with very little fines, so that the air spaces in the sack made it much lighter than one filled with finely pulverized ore. In a few instances that can be traced from the records the weights on certain claims changed noticeably at about the same time that new workings were being opened, so it is probable that changes in the character of the ore were the main factor in this diversity. On the Florence and the Great Western, where the ore usually ran fairly high in U_3O_8, and therefore was apt to be rather finely pulverized, the sacks averaged the heaviest, whereas on the Medea and the Vanadite, where the uranium content was almost uniformly low, the sacks averaged lightest.

VARIETIES OF ORES ENCOUNTERED.

The shipping ore from the Maggie C claim was for the most part a more or less coarse-grained sandstone impregnated with carnotite, which usually formed the binder between the individual sand grains, having probably replaced the original cement. The individual

grains of this sandstone were often thinly coated with fine carnotite, which gave the ore the appearance of being better than it actually was. Another variety found in many places on the Maggie C, and to some extent on the Florence and Nellie, called locally "rattlesnake ore," was a coarse-grained sandstone containing small patches one-quarter to one-half inch in size impregnated with black vanadium oxide and with carnotite, both kinds of patches being scattered at intervals of perhaps one-half to three-quarters of an inch in seemingly barren sandstone. The patches of carnotite were too small to separate by sorting, and the entire mass generally ran too low in uranium content to be shipping ore, so this ore was sent to the mill. Another type mined on the Maggie C was the brown or cocoa-colored ore previously described, which usually contained more than the average percentage of uranium; but as a large amount of the low-grade reddish-brown ore, chiefly iron bearing, was also found there, careful sorting and frequent tests in the electroscope were necessary to prevent shipping ore of too low grade. Very little high-grade, fairly pure carnotite was found on this claim—or on any of the other claims—and that found was mostly in stringers, in "bug holes," or in tiny pockets that were not more than 2 inches thick at the most, although some were 4 or 5 feet long. When such high-grade ore was discovered, the quantity was usually too small to permit sorting as a separate grade, consequently it was thrown into the shipping ore. One or two of these "bug holes" of high-grade ore were large enough to warrant keeping the ore separate, two sacks being obtained in all.

Such shipping ore as was found on the Henry Clay claim was invariably high in vanadium and was usually greenish-yellow to greenish-gray. In this ore the texture was rather fine, and it usually occurred in a finely-banded sandstone. The ore from the Florence was also of this banded or bedded type, being generally rather high in vanadium. A number of tons of ore from one pocket on this claim showed this bedded structure very plainly, much of it consisting of fine bands about one-sixteenth of an inch thick, of nearly pure carnotite, lying in a soft, almost claylike mass of black vanadium ore. A number of small "bug holes" of high-grade ore were discovered and mined on this claim, but no attempt was made to create a separate grade.

Most of the ore from the Nellie claim, adjoining the Florence, was similar to that from the Florence, except that no black ore with streaks of high-grade carnotite was found. Ore similar to that on the Maggie C, containing sandstone grains coated with carnotite, also occurred, but most of it was of finer texture than that on the Maggie C, and as a consequence usually contained a higher percentage of uranium.

On the Great Western claim the ore commonly contained a rather high percentage of calcite, and consisted of a fine-grained rather high-

grade shipping ore and a coarse-grained milling ore running much lower in uranium oxide.

On the Vanadite claim the ore was rich in vanadium, usually containing streaks of vanadic sandstone impregnated with carnotite between bands of gray or white barren sandstone.

The ores on the Medea claim, with one exception, were greenish and were found in a banded finely grained sandstone. In the exception mentioned the ore was rather coarse grained, closely resembling that on the Maggie C, and contained thin stringers of calcium vanadate and vanadic oxide,[a] rarely more than half an inch thick, which generally crossed the bedding of the sandstone and seemed to be small fissure veins.

On the Buckeye and Noontime claims the ore contained a high percentage of vanadium and was greenish to blackish in color. Some specimens from these claims were especially rich in vanadium and carried 1 to 2 per cent uranium oxide; they had a shiny, metallic luster and a bluish-green color. Such ore is locally called kentsmithite or mcmillenite.

THE ELECTROSCOPE.[b]

Radioactive ores accelerate the rate of discharge of electricity from a charged body because the rays they give off ionize the atmosphere surrounding the charged body. Hence an electroscope can measure radioactivity. The electroscope used by the National Radium Institute at the mines consisted of two cylindrical compartments of brass. The upper cylinder was about 3½ inches in diameter and was placed horizontally on the lower one, the ends being closed with mica windows to exclude disturbing air currents; the lower cylinder was placed vertically and closed at each end with a brass plate. Its lower face was flanged and screwed to a wooden base; on the upper face a short pipe supported the first compartment. A small brass rod, extending through a plug of sealing wax in this pipe that insulated the rod from the rest of the apparatus, was flattened at its upper end where it extended into the upper compartment and terminated at its lower end in a thin brass disk about 2½ inches in diameter, set at right angles to the rod and about half an inch from the top of the lower compartment.

Attached to the upper and flattened part of the rod and wholly within the upper compartment was a thin piece of aluminum foil 2 inches long and one-quarter of an inch wide, suspended from the top so that ordinarily it rested against the rod. When, however, the

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[a] Hillebrand, W. F., Merwin, H. E., and Wright, F. E., Hewettite, metahewettite, and pascoite; hydrous calcium vanadates: Proc. Am. Phil. Soc., vol. 53, 1914, pp. 31-54.

[b] For a more complete description of the electroscope and electroscopic methods see: Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium, Bull. 70, Bureau of Mines, 1913, pp. 64-66; Lind, S. C., and Whittemore, C. F., The radium-uranium ratio in carnotite, Tech. Paper 88, Bureau of Mines, 1915, pp. 17-19; Lind, S. C., Practical methods for the determination of radium; interchangeable electroscope and its use, Jour. Ind. and Eng. Chem., vol. 7, 1915, p. 406.

leaf and rod were charged with electricity (generated on a vulcanite rod rubbed against wool or silk) the leaf was repelled from the rod and extended at an angle from it. If the instrument were then left undisturbed the leaf would very slowly fall to its former position as the charge gradually leaked away. The rate of this fall (observed through a microscope attached to the apparatus by means of a brass arm), measured by a stop watch in terms of the divisions on a scale in the eyepiece of the microscope, was termed the "natural leak" of the instrument. If the leaf were again charged and a quantity of carnotite put in the bottom compartment through a door provided for that purpose, the rate of fall would increase.

The rate of such fall is, within reasonable limits of accuracy for a field test, always proportional to the amount of uranium oxide in the carnotite. Therefore, if this sample of ore be removed and the same quantity of a standard ore of known uranium content be put in the machine and the rate of fall of the leaf be again measured, the uranium oxide content of the sample can be computed by the following formula: (Rate of fall with sample minus rate of natural leak) : (Rate of fall with standard minus rate of natural leak) : : (Per cent U_3O_8 in sample) : (Per cent U_3O_8 in standard).

PRECAUTIONS IN USING ELECTROSCOPE.

In using the electroscope several precautions are necessary, perhaps the most important of which is to have the ore ground to the same fineness. Experiment has shown that the influence the ore exerts, and hence its apparent uranium content, varies greatly with the fineness of the ore. This error has been found to be as high as 100 per cent when comparing ore ground to pass a 200-mesh screen with similar ore passed through a 30-mesh screen, the finer material giving the lower result. For this reason the ore to be tested and the standard ore must be passed through the same mesh of screen, and it is obvious that in grinding the samples all of the material must be passed through the screen; otherwise, the lower-grade material being the harder, a partial concentration would vitiate the results. The screen used was about 30-mesh bolting cloth stretched on a wooden hoop, but 40-mesh or even 60-mesh screens could be used. The same screen should be used for both sample and standard, and should be thoroughly cleaned after each test. The quantity of ore, both standard and sample, placed in the electroscope should be constant.

The error resulting from using varying quantities of ore is not as great as that resulting from a wide difference in fineness, but is sufficient to destroy accuracy. For this purpose the electroscope used at the mines was equipped with a small brass disk, fitting into the bottom compartment, that had a raised rim about one-sixteenth of an inch high. When the ore was spread over this disk and any excess scraped off with a straight metal blade, such as a common

table knife, there remained a film of ore one-sixteenth of an inch thick and the diameter of the disk. Care was taken to have the surface left perfectly smooth in order to insure that the same amount was always retained on the plate. When testing finely powdered material, such as concentrates, the required smoothness was obtained by pressing the material down with the flat underside of a second disk, using a slight rotary motion. The plate was then carefully pushed through the door onto the floor of the lower compartment. As illumination on the leaf should be constant for both readings, it is usually more convenient to make the electroscopic tests at night and by artificial light, a candle back of the electroscope throwing the leaf into sharp silhouette.

PACKING, STORAGE, AND TRANSPORTATION OF ORE.

PACKING AND STORING ORE.

About 1 per cent of the total mining time was employed in packing and loading ore. The freighting of ore, which was done by team and wagon, was not begun until late in August, 1914. During the winter months the sacked ore from the Florence, Nellie, Great Western, and Vanadite claims was packed by a train of two to eight burros, two being owned by the company and the others hired from miners, to the warehouse on the Maggie C claim. The Maggie C ore was also stored there during the winter, and the freighters loaded from that point the shipping ore from the claims mentioned. Once, in December, 1914, it was necessary to pack the Medea ore to the main road with burros, but at other times the teams were able to load on the claim.

A table showing the amount of time spent in packing ore and loading follows.

Percentages of total mining time per month spent in packing ore and storing in warehouse.

Month.	Maggie C.	Florence.	Nellie.	Great Western.	Henry Clay.	Medea.	Buckeye and Noontime.	Dixie.	Vanadite.	Uranite.	Total.
1914.											
June	0.0	......	......	0.0	0.0	......	......	......	......	......	0.0
July	.0	0.0	0.0	.0	.0	......	......	......	......	......	.0
August	.0	.0	.0	.0	.0	0.0	......	......	......	......	.0
September	.1	.1	.5	.5	.0	.0	0.0	......	0.0	......	.75
October	.0	.04	.04	.3	.0	.0	.6	0.0	.0	0.0	.98
November	.0	.0	.0	.09	......	......	......	.0	.36	.0	.45
December	1.8	.09	.14	.36	......	.05	......	......	.18	.0	2.62
1915.											
January	1.9	.0	.2	.1	......	.0	......	......	......	......	2.2
February	2.06	.14	.3	.4	......	.0	......	......	......	......	2.9
March	.8	.3	.0	.0	......	.0	......	......	......	......	1.1
April	.04	.6	.0	.36	.0	.0	......	.0	.0	.0	1.0
May	.0	.19	.05	.15	......	......	......	.0	.0	......	2.9
June	.0	.2	.05	.25	.0	......	.0	.0	......	......	.3
July	.0	.66	.04	.0	.0	.0	......	......	......	......	.7
August	.0	.58	.02	.13	.0	.0	......	......	......	......	.73
September	.0	.10	.10	......	.0	.0	......	......	......	......	.20
October	.0	.22	.11	.33	......	.0	.0	......	......	......	.66
Average	.4	.2	.1	.2	......	......	......	......	......	......	[a]1.0

[a] Approximately.

TRANSPORTATION OF ORE.

The distance between the claims and the nearest railroad station, Placerville, Colo., was approximately 58 miles, and the teams made the round trip in seven days. Four or six horses hauled two wagons, one of them a trailer. Four-horse teams hauled 3½ to 4 tons of ore; the six-horse teams 5 to 5½ tons. As Long Park is isolated, all commissary, mine, and mill supplies, including lumber, machinery, etc., were hauled from the railroad station at Placerville or intermediate points. Usually the freighters who hauled the ore loaded the wagons, but frequently were assisted by miners detailed for this work. At Placerville the ore was unloaded into the warehouses rented by the institute, or loaded direct into the railroad cars, which were narrow-gage cars holding approximately 35,000 pounds each. Two cars were shipped at one time, and were reloaded into one broad-gage car at Salida, Colo.

MISCELLANEOUS AND GENERAL WORK.

MISCELLANEOUS.

Under "miscellaneous work" the most important item was cutting and setting up mine timbers, but it also included making rough roads and trails and carting tools to the various workings, setting up and moving sorting tables, and similar unclassified work. Not more than half of 1 per cent of the total mining shifts was spent in these tasks. Roughly, half of the time under this item is directly chargeable to the Maggie C, one-fourth to the Florence, one-tenth to the Great Western, and the rest to the other claims.

GENERAL EXPENSE.

Under the caption "general expense" were included items incidental to mining, but not directly chargeable to any one claim in particular. Chief among these is blacksmithing. Approximately 4 per cent of the total mining time was charged to the "general" account, which included, in addition to blacksmithing, such items as making tool boxes, repairing blacksmith shop, constructing powder magazine and warehouse, grinding samples for electroscopic tests, overtime for loading wagons, etc. The blacksmithing time, rightly chargeable to the mine account, averaged a little less than half a shift daily, or something under 2½ per cent of the total mining time. The item included sharpening drill steel and repairing tools, and also work in mill construction, laying track on the Maggie C (listed separately under the special table for that claim, see page 28), and camp repairs. The distribution of the blacksmith's time, based on the number of holes drilled per claim, is as follows: Maggie C, 50.5 per cent; Florence, 22.9 per cent; Nellie, 6.6 per cent; Great Western, 8.7 per

cent; Henry Clay, 2.6 per cent; Medea, 3.9 per cent; Buckeye and Noontime, 1 per cent; Dixie, 1.3 per cent; Vanadite, 1.5 per cent; Uranite, 1 per cent. An average of about eight steels were sharpened per hour of the blacksmith's time, at a cost of 3½ cents per hole blasted. The figures for the Buckeye and Noontime claims are low, because during the 1914 work on these claims the tools were sharpened by the miners.

Other work not included in the foregoing is the time of the mine foreman, which was distributed over the various claims as follows: Maggie C, 56 per cent; Florence, 11½ per cent; Nellie, 5½ per cent; Great Western, 11 per cent; Henry clay, 3½ per cent; Media, 2½ per cent; Buckeye and Noontime, 2¾ per cent; Dixie, 1½ per cent; Vanadite, 1½ per cent; Uranite, 1 per cent; camp, 1½ per cent. These figures take into consideration the number of men at work on the claim during each month of the mining period, the distance of the claims from camp, the time necessary to make the circuit, and the fact that at times several workings were visited in one trip of inspection, and at others long rides were necessary to visit one isolated miner. The figure for camp includes supervision of the building operations, of repairs and general maintenance, and is rightly chargeable to commissary, but was inserted here to indicate the correct ratio.

COMMISSARY AND CAMP.

The distance of the mines from any town necessitated living quarters for the men, consequently a camp (see Pl. VI, *A*) was established at the extreme western end of Long Park, on the Maggie C mill site and convenient to a small spring. The kitchen, the foreman's office, and the guest house were frame buildings 10 by 14 feet in dimension. The mess room and the men's quarters were tents stretched over a frame made of 2 by 4 inch lumber, but provided with wooden floors and wooden sidewalls built up about three feet from the floor, where the tent walls were attached. The mess tent, 14 feet wide and 16 feet long, was placed at the rear of the kitchen building and connected with it by a door, another door gave access from the outside. The tent had windows on three sides, and contained two tables that could accommodate 30 persons. In cold weather it was heated with a wood-burning heating stove, in summer it was well screened. The men were housed in six 12 by 14 foot tents accommodating five men each, a similar tent being provided for the cook and flunkeys. The men slept in wooden bunks. In winter the wooden sides of the tents were covered with heavy tar-paper and banked with earth; air-tight stoves made the quarters comfortable. Wood for these stoves was hauled to camp by team on the institute's time and piled conveniently for cutting. Each tent was equipped with a kerosene lan-

A. CAMP AT LONG PARK, COLO.

B. CLOSE VIEW OF CONCENTRATING MILL AT LONG PARK, COLO.

tern, a wash basin, bucket, and broom. The men were given individual canvas water bags for drinking water, which they took with them daily to work. For the superintendent a 10 by 14 foot "Colorado tent house" was furnished. The cook house, foreman's and superintendent's offices, and guest house were provided with gasoline lamps. In the fall of 1914 a telephone was installed.

The saddle horses, which numbered one to three during the work, were housed in a brush corral and a small frame stable. A chicken house adjoined the corral.

The National Radium Institute maintained its own commissary, charging the men $1 a day for board, and not seeking to make any profit from this charge. The food comprised canned goods and other provisions, fresh meat, vegetables, and fruit in season. A plentiful supply of pure water was always kept on hand, although during more than half of 1915 this had to be hauled daily by team from a spring a mile and a half distant. Two roomy cellars housed the provisions and kept them cool and dry; heavy double doors prevented freezing in winter. The meat house was thoroughly screened and sheltered.

Many of the men working on the more distant claims did not return to camp for the midday meal and a special attempt was made to have their lunches attractive and appetizing. As a result of the attention given to the commissary the men were satisfied, and changes in the personnel were infrequent. Of 19 men employed August 1, 1914, 13 were still on the payroll on October 1, 1915, fourteen months later. As breaking in new men to carnotite mining is expensive, because of their inefficiency for the first two weeks and the increased demands on the foreman in educating them to know the ore, the policy of allowing no grounds for dissatisfaction with the commissary paid splendid dividends, even though the department might be conducted at a slight loss. The superintendent ordered all supplies, buying monthly and in bulk by means of competitive bids, inventories being taken at intervals of not more than a month.

For amusements, through the kindness of Drs. Douglas and Kelly of the National Radium Institute, the men were provided with a baseball outfit and a croquet set. Some of the men played at bowling or "bocci." A phonograph was in the foreman's house, which also served the purpose of general club house and reading room. All gambling was forbidden.

SAFETY AND HEALTH PRECAUTIONS.

Mining for carnotite is less hazardous than many other types of mining, because the workings are relatively small and shallow and largely open to the day, and also because the miners must possess a rather high grade of intelligence. Safety in working conditions and

prevention of accidents received foremost consideration, the rule of "Safety first" being strictly enforced, as far as possible, and constantly impressed on the men.

The chief dangers were from possible falls of rock and the handling of explosives. Each miner was warned repeatedly to take time to make the roof safe, and always to examine the roof and the walls carefully after each blast. As the men were paid by the shift, there was no particular difficulty in enforcing this precaution; yet the only accident that occurred was caused by violation of this rule, a miner being slightly injured by the fall of a piece of rock he had neglected to pick down.

The foreman was always on the lookout for loose rock or places needing support. The superintendent never went into a tunnel without carefully examining the walls and roof, or into a deep surface excavation without examining the walls. It was repeatedly impressed on the miner that whenever the roof was not safe, his first duty was to pick down all loose rock and make it so, or if timbering was necessary, to get timbers and put them in place before undertaking any other work. As explained previously, on a claim where only one or two men worked the timbers were cut by the miner as needed, whereas on claims where a number of men were working a man was detailed periodically to cut timbers, which were then hauled by team to the miners' working place. As a further precaution against falls of roof the stopes were refilled with waste as soon as possible after excavation.

The handling of explosives was carefully supervised. The main supply of dynamite, rarely more than a ton at any one time, was kept in a dugout in a hillside. The dugout had a rock wall in front, a thick roof of timbers and earth, and was always kept locked, the mine foreman having the key. This magazine was dry, cool in summer and comparatively warm in winter, and sufficiently out of the way so that it was not visited except to obtain dynamite. As far as possible the dynamite was taken to the individual claims in the original unopened boxes, but where only a small quantity was required it was transported carefully in a canvas ore sack. The supply of fuse and caps were kept in a separate place under lock and key, and the small amounts for daily use at the claims were kept separate from the daily supply of dynamite. The miners were never permitted to handle dynamite carelessly. In cold weather hot-water powder thawers were supplied. Cap crimpers with wide crimping faces were furnished and their use was compulsory. In loading blast holes only wooden tamping sticks were permitted, a supply being kept on hand and issued as needed. As previously stated, the minimum length of fuse permitted was 2 feet, giving the miner about a minute and a half to seek shelter after igniting the fuse.

Before blasting, the warning cry of "fire" was always given several times and the fuse was not lighted until the rest of the miners in the vicinity had taken shelter. Misfires were rare, except in very cold weather, when the fuse, in spite of repeated warnings, was sometimes handled too roughly before it was thoroughly warmed. The miners had orders in case of a misfire to wait a safe interval of time before approaching the hole.

Intoxication, often a contributory cause of accidents in mining, was unknown, as the nearest saloon was at Placerville, Colo., 60 miles distant, and it was understood that any man discovered with liquor in his possession would be discharged immediately.

No sickness more serious than a slight cold occurred during the mining work, one contributing cause to this exemption being the supply of pure drinking water from local springs. During the first few months of the work the spring on the Maggie C mill site produced about two barrels of water a day, enough for domestic purposes, but in the dry season in 1915, the spring supplied just enough water for table use, and water for cooking and for the men to carry to their work had to be brought from a spring about a mile and a half distant. As the country above these springs was wild and uninhabited, the purity of the drinking water was assured. A simple, wholesome diet, as varied as possible, was supplied by the camp commissary. Empty cans and other refuse were carried some distance from the camp and buried. Chloride of lime was used freely as a disinfectant.

In addition to these precautions, literature on mine safety was liberally supplied to the miners.

PRODUCTION OF CARNOTITE ORE BY NATIONAL RADIUM INSTITUTE.

The table following gives data on all shipping-grade ore mined from the institute's leased claims prior to October 31, 1915, and represents 49 narrow-gage carload lots which were shipped to Denver, Colo. There were mined approximately 970 tons (gross weight) of ore, the net weight after sampling was approximately 960 tons, and the net dry weight, after deducting about 27¾ tons of moisture, was approximately 932 tons of ore averaging 2.6 per cent U_3O_8 content. In mining this shipping ore, about 2,000 tons of milling ore were produced having an average content of 0.8 per cent U_3O_8.

Formerly this low-grade ore was wasted in mining shipping ore, and as the recovery by concentration of the milling-grade ore was about 60 per cent, about 2,300 milligrams (theoretical content) of radium element were recovered by concentrating material which, in former years, was lost.

After the mining described in this paper, approximately 60 tons of shipping ore with an average content of 2½ per cent U_3O_8 were produced during assessment work for 1916, the total production of

shipping ore being, therefore, approximately 990 tons. Also, there was produced in 1916 from low-grade milling ore approximately 300 tons of carnotite concentrates with a U_3O_8 content averaging 3 per cent. The shipping ore contained approximately 51,000 pounds of uranium oxide and the concentrates about 18,000 pounds. Thus the total production of ore and concentrates represents approximately 69,000 pounds of uranium oxide, or its equivalent of about 8.8 grams of radium element.

Shipments of carnotite ore mined by the National Radium Institute from its leased claims up to October 31, 1915.

Car lot No. [a]	Number of bags.	Net weight after sampling.	Moisture.	Weight of water.	Net dry weight.	U_3O_8 content.	V_2O_5 content.	Weight of U_3O_8 in ore.	Weight of V_2O_5 in ore.
		Lbs.	*Per ct.*	*Lbs.*	*Lbs.*	*Per ct.*	*Per ct.*	*Lbs.*	*Lbs.*
1	900	64,767	2.25	1,457	63,310	2.26	5.43	1,430	3,437
2	1,100	79,075	2.08	1,644	77,430	2.71	4.45	2,098	3,445
3	1,063	73,688	2.00	1,473	72,215	2.81	4.06	2,029	2,931
4	1,096	79,160	2.40	1,900	77,260	2.83	3.87	2,186	2,989
5	1,109	78,884	2.00	1,577	77,307	2.44	3.46	1,886	2,674
6	1,091	78,145	3.00	2,344	75,801	2.21	4.43	1,675	3,357
7	1,099	80,865	6.60	5,337	75,528	2.32	4.40	1,752	3,323
8	1,042	70,545	5.20	3,668	66,877	2.76	3.77	1,845	2,521
9	1,199	82,625	5.40	4,462	78,163	2.31	4.40	1,805	3,439
10	1,154	78,925	3.60	2,841	76,084	2.24	3.83	1,704	2,914
11	1,103	76,635	4.20	3,218	73,417	2.56	4.21	1,879	3,090
12	1,238	82,625	2.70	2,230	80,395	2.53	5.04	2,033	4,051
13	1,186	82,645	3.00	2,479	80,166	2.85	3.64	2,284	2,918
14	1,092	73,510	4.09	3,007	70,503	2.37	4.16	1,670	2,932
15	1,205	83,880	1.90	1,593	82,286	2.97	3.97	2,443	3,266
16	1,171	79,065	2.63	2,079	76,986	2.98	4.44	2,294	3,418
17	1,158	78,130	1.33	1,039	77,091	2.53	3.64	1,950	2,806
18	1,244	82,810	1.41	1,167	81,643	2.61	4.02	2,130	3,282
19	1,168	79,935	2.55	2,038	77,897	2.89	5.07	2,251	3,949
20	1,177	81,705	2.67	2,181	79,524	2.51	4.13	1,996	3,284
21	1,102	78,945	2.52	1,989	76,956	2.69	5.15	2,070	3,963
22	1,090	74,475	3.00	2,234	72,241	2.63	5.55	1,899	4,009
23	1,102	79,999	1.99	1,592	78,407	2.37	4.78	1,858	3,747
24	1,056	75,425	1.49	1,124	74,301	2.85	4.46	2,117	3,313
25	586	43,081	2.30	991	42,090	2.52	4.59	1,060	1,931
Total	27,531	1,919,544		55,664	1,863,878			48,344	80,989
Average [b]			3.00			2.60	4.30		

[a] Lots 1 to 24 represent broad-gage railroad cars; lot 25 a narrow-gage car.
[b] Approximate.

CAUTION IN BUYING CARNOTITE ORES.

Caution is necessary in buying carnotite and other radioactive ores. Such ores have always been purchased on the basis of their uranium content, as determined by chemical analysis. This method ordinarily is sufficiently accurate as indicating the equivalent radium content of the ore. In general, and especially where ore is purchased from unknown sources, the method of determining the radium content of an ore from the chemical analysis for its uranium content is not satisfactory, especially if the ore is purchased with the prime object of extracting the radium.

Several instances are known to the writers where an admixture of yellow sodium uranate and other uranium compounds, which were

ground to the size of some of the grains of ore, was added. The radium from such material had been extracted, and as the sodium uranate and other uranium compounds usually have a high content of U_3O_8 it can be readily seen thát a small quantity added to an ore is sufficient to increase the price of the ore considerably, and if such an ore is bought on the basis of its U_3O_8 content, and no special radium determination is made, serious losses will result to the purchaser. The admixture of small quantities of uranium compounds can not be easily detected by the unaided eye, but can be readily seen under the microscope when a careful examination of the material is made. Radium determinations should be made, therefore, on such ores by means of the electroscope.

Some carnotite ores contain considerable calcium sulphate and other sulphates which interfere with the extraction of uranium, vanadium, and radium by some processes. Such ores principally come from Utah, but are found in various parts of Colorado. Hence, carnotite ore should always be tested for its sulphate content.

Chapter II. GRINDING AND SAMPLING OF SHIPPING ORE BEFORE CHEMICAL TREATMENT.

During the early stages of the mining operations the National Radium Institute had its ore ground and sampled by other firms. As experience soon showed that this work could be done better and more cheaply in a special plant, the institute installed its own grinding and sampling plant at Denver, Colo.

The principal difficulty in crushing, grinding, and sampling carnotite ore is that much of the fine light dust produced can not be satisfactorily saved in plants constructed primarily for sampling ores that produce less dust, or heavier dust, than carnotite ore. This fine dust, being rich in carnotite, contains more uranium, radium, and vanadium than the coarser particles of the ground ore. Unless all of the machinery used in the crushing, grinding, and sampling is made practically dust proof and is provided with dust-collecting devices, much of this fine rich dust, some of which is as light as smoke, is lost, and a true sample of the ore is difficult to obtain. The problem of saving the dust, not only to prevent waste but also to obtain a true sample, had to be especially considered in constructing the sampling unit. The entire unit was made practically dust proof, and ample provision was made for collecting and sampling the dust.

DESCRIPTION OF CRUSHING, GRINDING, AND SAMPLING PLANT.

The hand-sorted shipping-grade ore as it came from the mines consisted of lumps that would pass through rings 1 to 3 inches in diameter, mixed with finer material. The coarse ore, previous to chemical treatment, has to be ground, and at first the ore, after having been crushed to ¼-inch size, was ground to pass a 40-mesh screen. As the capacity of the mill could be increased considerably by coarser grinding, tests were made to determine whether ore more coarsely ground could not be treated equally well. These tests proved that coarser grinding was an advantage, as it facilitated filtering. Thereafter all ore was reduced to pass through a 10-mesh screen placed at an angle of 45° and giving a product per screen test as follows:

Results of screen test of ground ore as delivered to the chemical plant; material passing 10-mesh screen placed at 45° angle.

	Per cent.
On 14 mesh	2.0
On 20 mesh	9.4
On 28 mesh	15.0
On 35 mesh	34.5
On 48 mesh	19.0
On 65 mesh	11.2
On 100 mesh	5.4
On 150 mesh	2.0
On 200 mesh	.6
Through 200 mesh	.9
	100.0

NOTE.—All dust collected passes through a 200-mesh screen.

As the ground material was easy to sample, the sampling device was inserted at that point. This device was so arranged that one four-hundredth of the entire weight of ore treated was cut and collected as a representative sample. (See p. 60.) This sample was mixed with an equal proportion of the dust collected, quartered, split and a final sample taken. The moisture sample was obtained in the usual manner just before the ground and sampled ore was weighed, as described on page 60.

DUST COLLECTION AND REMOVAL.

All connections between the individual apparatus were of strong galvanized-iron pipes and spouts, the joints being soldered. The elevator housing was of galvanized iron with all joints thoroughly packed. The screen and the sampling device were entirely inclosed. Access to the elevator and the sampling device could be had through close-fitting doors that were packed, hand screws being provided to hold them in place.

Tests made to determine what the loss would be without a fan and dust-collecting device showed that the loss of dust would be enormous; the dust from the crusher and the rolls made it almost impossible for anyone to remain in the grinding room, and the air currents created by the elevator and other moving parts carried the dust through all openings.

The dust from carnotite ore has a relatively high uranium content and is very irritating to the nose, throat, and lungs. Therefore health considerations alone necessitated the installment of fans and dust-collecting devices. The value of the escaping dust was considerable. For instance, the dust collected from a carload of 80,000 pounds of ore amounted to 1,500 pounds or about 2 per cent. In winter, when the ore arrives from the mines very moist, the amount would be less. This dust, however, averages about 8 per cent

uranium oxide, whereas the ore treated averages 2.5 per cent, so that the loss of carnotite in the escaping dust averages about 6 per cent. Of course, not all of the escaping dust was lost before the dust-collecting system was installed as some of it, after settling, could be collected from the floor and other places in the mill. The dust also works into the bearings and other wearing parts of the machinery.

Dust-collecting devices being an absolute necessity as regards safety, health, and economy, a Sturtevant "Monogram" No. 3 blower and a tubular dust collector were installed. The fan was run to give a capacity of about 1,250 cubic feet of air per minute. The dust collector consisted of two circular compartments each having 20 round openings 8 inches in diameter, to which cotton tubes or hose were attached. The lower compartment ended in a cone from which the collected dust could be drawn through a gate shutter. The 20 tubes, which were 16 feet long, practically filtered the dust from the air, the latter escaping through the meshes of the cloth. As some of the finest dust escaped through the meshes, the entire 20 tubes were surrounded by a sheet of cotton cloth, which was fastened to the outside wall of the upper and lower compartments. Thereafter practically all the dust was collected and saved. After the sampling of each carload of ore, the dust was drawn from the collector, and the entire unit thoroughly cleaned.

The fan, placed on the upper floor of the plant, was connected with the crushing and grinding apparatus by means of galvanized-iron pipes. Two 4-inch pipes were soldered into the upper side of the discharge spouts of the crusher and the rolls, respectively, and joined about 3 feet above the intake into one 7½ inch vertical pipe bent gradually to a right angle where it connected with the fan intake. This pipe was also connected with the intake spout of the elevator boot. The dust drawn into the fan was blown through a pipe into the upper compartment of the collector. Various other pipe connections provided, such as a dust intake pipe near the top of the elevator and one connecting with the screen housing were not necessary. Most of the dust from the crusher and the rolls was removed through the pipes connected to discharge spouts. Any dust which was not thus removed and was carried to the lower part of the elevator by the draft from the descending buckets was sucked into the spout connected with the fan.

OPERATION OF PLANT.

The plant was belt driven from a 25-kilowatt Westinghouse motor. One man operated the entire plant, dumped the ore into the ore bin, looked after the oiling and repairs, and sacked the ground ore. The final sampling was also done by him under the supervision and in

the presence of one of the Bureau of Mines employees. The capacity of the plant was approximately 10 tons per 8-hour shift.

The plan and sections of this mill are shown in Plates VII to X.

GENERAL PROCEDURE.

The ore was unloaded from the railroad cars, elevated to the upper floor of the grinding plant, and emptied into a bin holding about 25 tons. Through an opening 1 foot square in the ore bin *1*, the ore was mechanically fed through a swing-gate feeder, *2*, over a chute into a 5 by 9 inch Sampson crusher, *3*, set to crush the ore to ¼-inch size. The crushed ore dropped through a spout, *4*, into the boot of a bucket elevator, *5*, and was then elevated to the upper floor in the building, where it was discharged into the feeder, *6*, of a Newago 10-mesh vibrating screen, *7*, set at an angle of 45°. The oversize from the screen fell into a feed pipe, *8*, and dropped into the hopper of a set of 12 by 12 inch Davis rolls, *9*, on the lower floor of the building. The rolls were set close enough to make a product passing the Newago screen, and this product dropped through a spout, *10*, into the elevator boot, *5*. The material elevated to the upper floor of the building discharged into the feeder, *6*, of the vibrating screen, *7*. The oversize was returned by gravity through spout *8* to the rolls, *9*.[a] The undersize from the screen was fed by gravity through spout *11* into the sampling device, *12*, which was suspended from the upper floor of the building. The reject from the sampling device dropped through a spout into the ore bin, *14*; the sample, representing about one four-hundredth of the quantity of ore treated, was collected in a small sample bin, *18*. Both the ore bin for collecting the reject, and the sample bin are made of heavy galvanized iron and are dustproof. A flow-sheet of the process follows:

[a] Only one elevator was installed for handling the products from the crusher and the rolls.

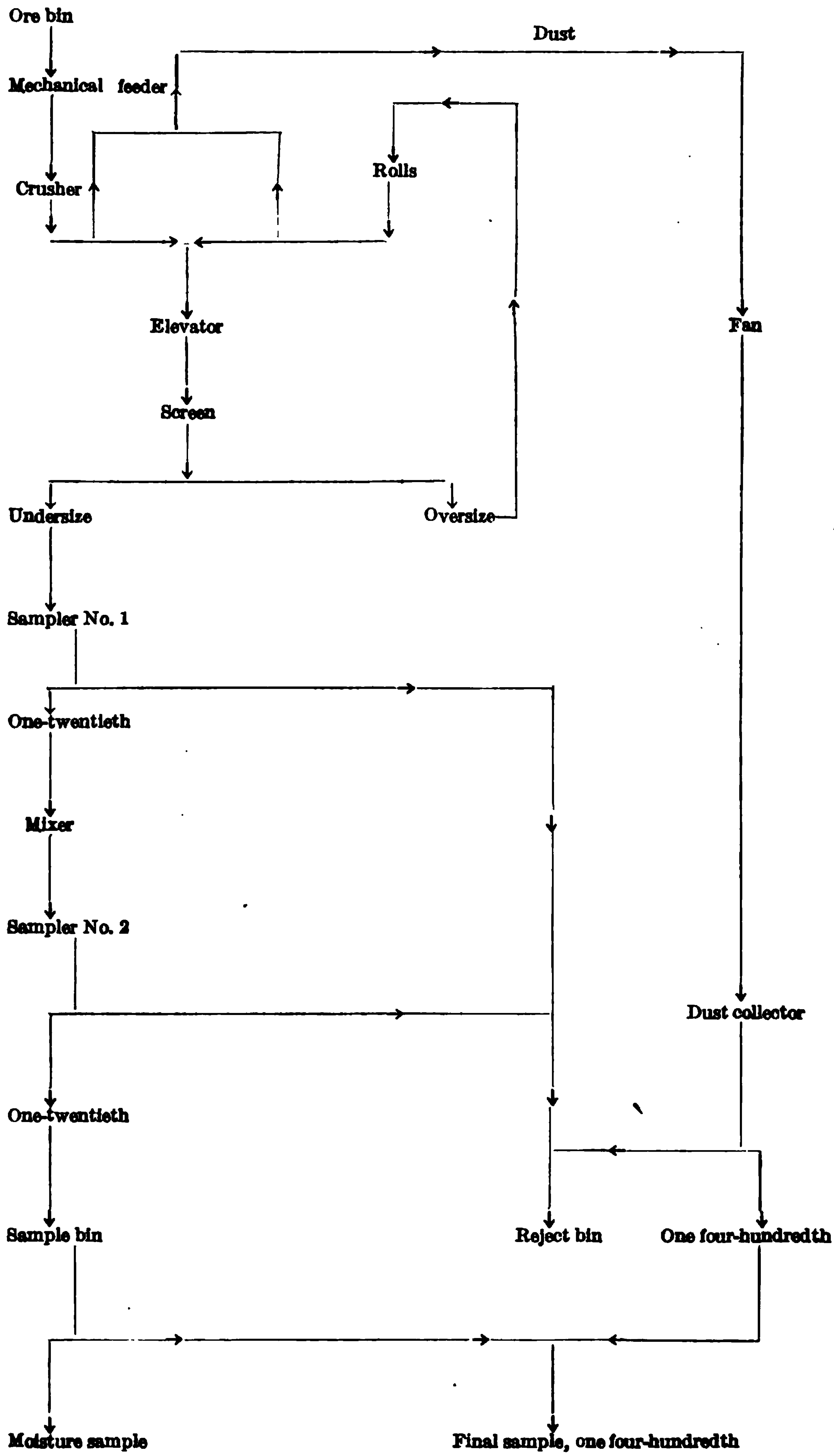
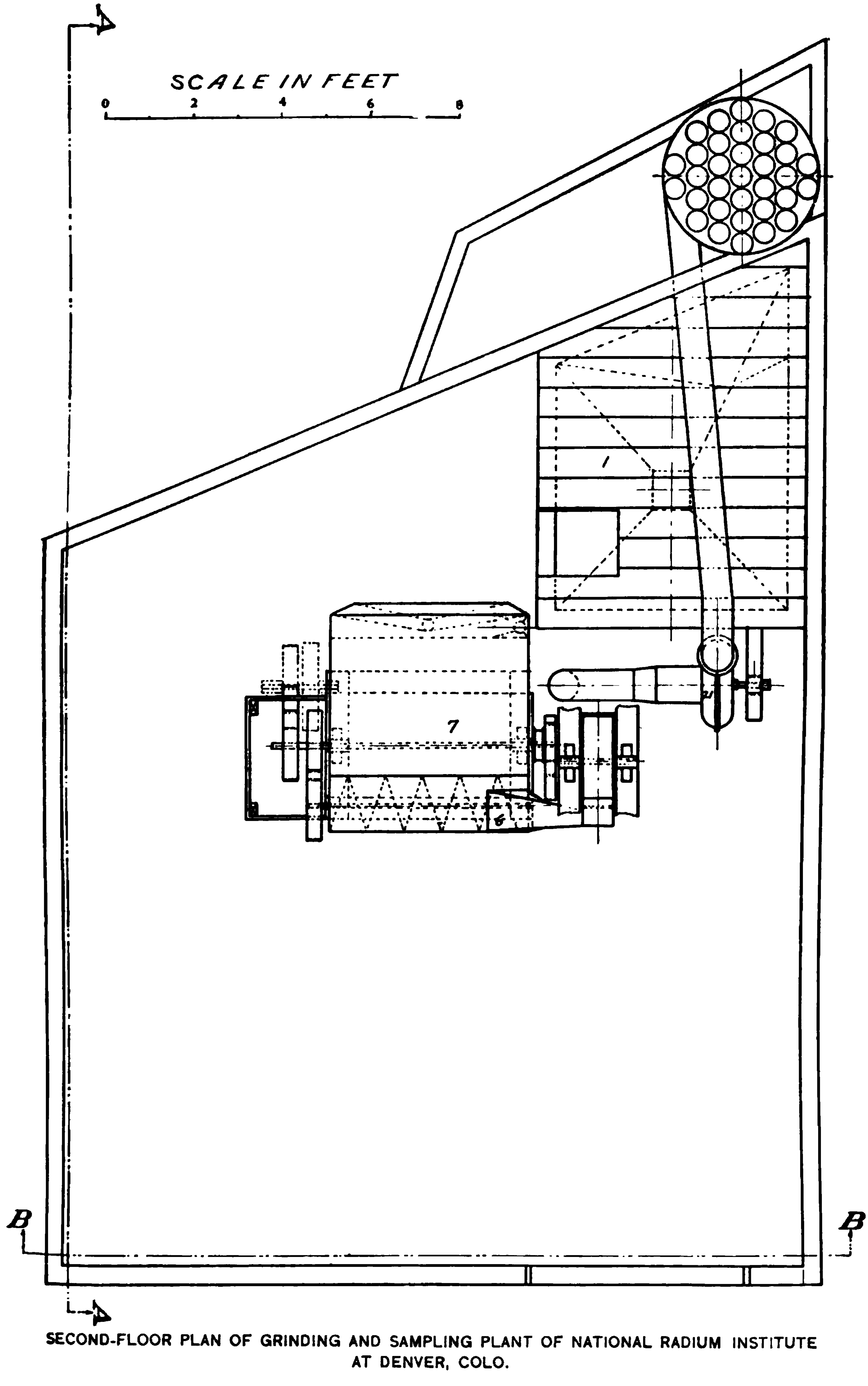
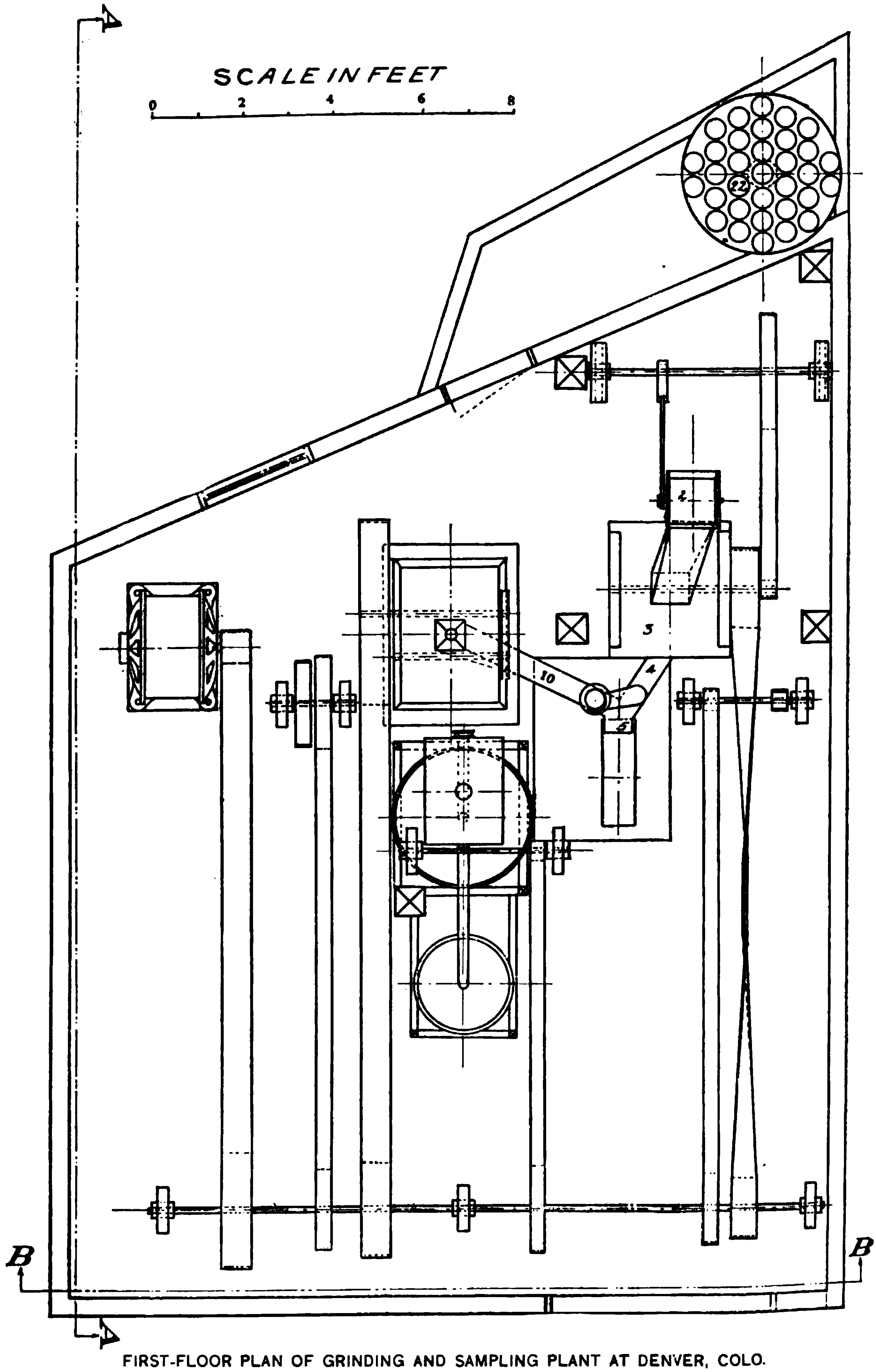


FIGURE 3.—Flow sheet of process of crushing, pulverizing, and sampling carnotite.



SECOND-FLOOR PLAN OF GRINDING AND SAMPLING PLANT OF NATIONAL RADIUM INSTITUTE AT DENVER, COLO.



FIRST-FLOOR PLAN OF GRINDING AND SAMPLING PLANT AT DENVER, COLO.

MECHANICAL SAMPLING DEVICE.

The mechanical sampling apparatus consisted of two Snyder samplers with a mixer between. These samplers were cast-iron pans 16 inches in diameter with flaring sides, set edgewise on a horizontal revolving shaft. A spout projecting through the flaring side of the pan passed under the feed spout at each revolution of the machine and cut and delivered a sample.

As shown in figure 4, the operation was as follows: Spout *b*, projecting through the flaring side of pan *a*, passed at each revolution under feed spout *c* and delivered a sample into the spout *d* of the

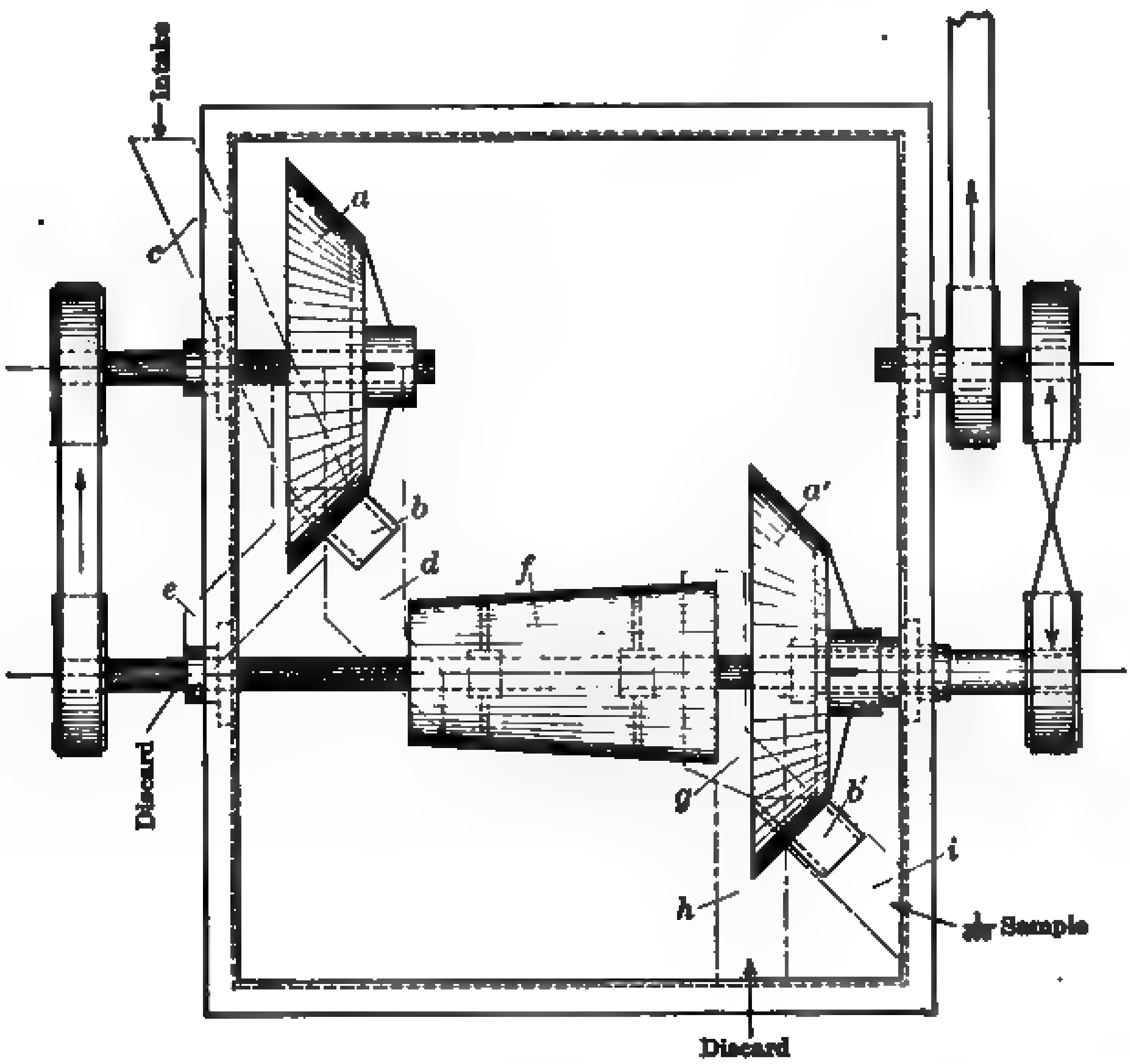


FIGURE 4.—Sampling device.

revolving mixer *f*. During the remainder of the revolution of the pan *a* the stream of ore from feed pipe *c* was diverted as reject into spout *e*. The cut sample from the first pan passed through spout *d* into the mixer *f*, and the mixed material was discharged therefrom into feed spout *g*. The stream of the sample passing from spout *g* was again cut by the projecting spout *b′* of the second pan *a′* at each revolution, the cut sample passing into spout *i*, which was connected with the sample bin below. The stream of ore during the remaining part of each revolution was diverted by the flaring side of pan *a′*, and was

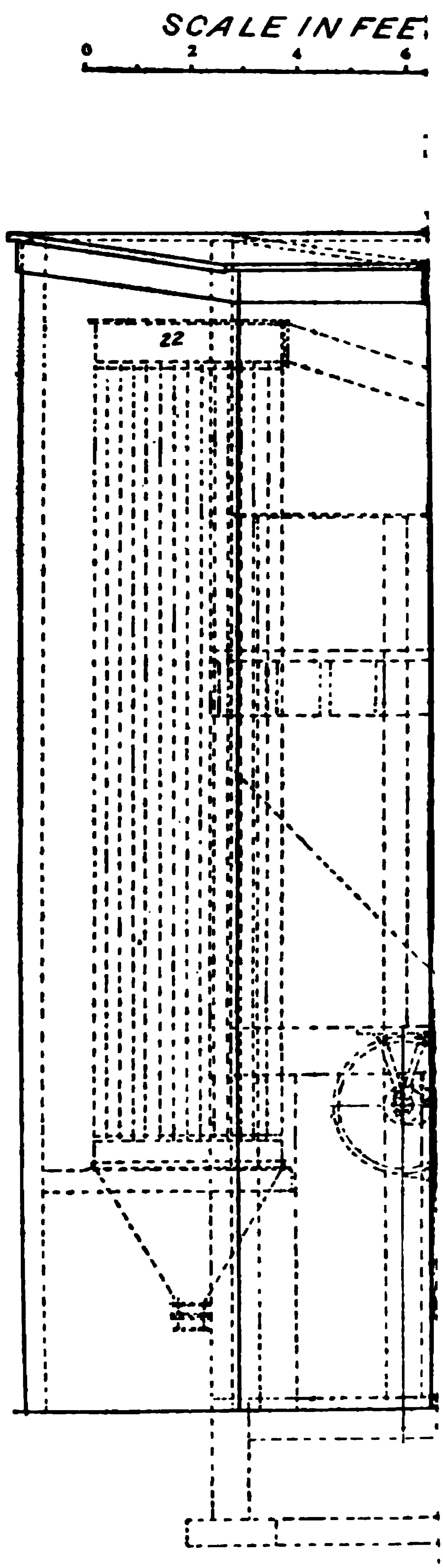
delivered as reject through pipe *h* into the ore bin below. Pipes *e* and *h* were connected with the same ore bin.

The entire sampling machine was inclosed in a dust-proof box made of galvanized iron 28 by 20 inches, by 45 inches high. One side of the housing could be removed for inspection of the machine; the other contained a door. The removable side was packed with felt, and the door was fastened to the box by means of hand screws.

The opening of the spout in the pan was 1¾ inches. One-twentieth of the entire quantity treated was cut by the first pan, the second pan also cut one-twentieth of the sample, the final sample obtained being one four-hundredth of the total ore passed through the machine.

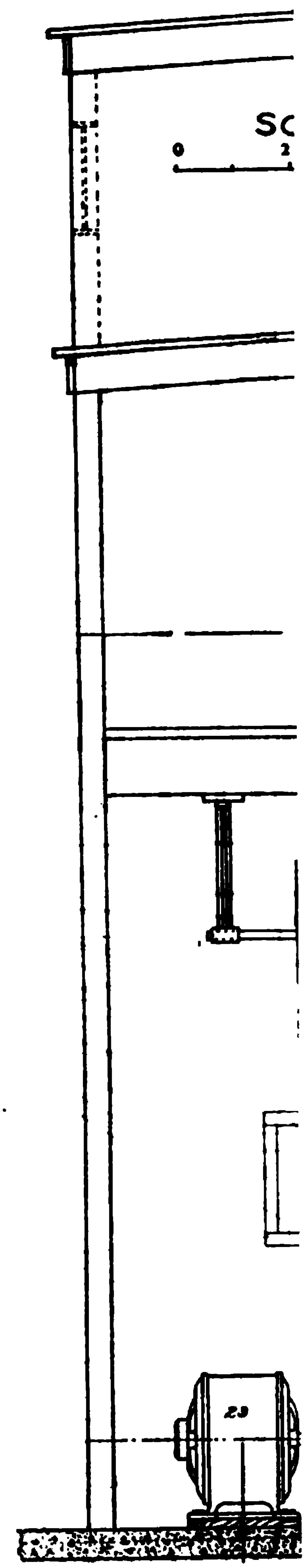
COLLECTION OF MOISTURE SAMPLES.

As soon as an entire carload of shipping ore had been ground and mechanically sampled, and just before the ore was weighed, the moisture sample was taken from the ore collected in the sample bin. As both the rejected ore from the sampler and the cut sample were kept in air-tight galvanized-iron bins under like conditions, the moisture sample was taken from the mixed sample (see p. 55) before the final cutting was made. In the early stages of plant operation the moisture samples were taken before the ore was dumped into the ore bin of the sampling plant. By careful tests the authors found that the moisture content of the ore varied slightly during the sampling, the ore, which is extremely hygroscopic, losing or gaining moisture during the process. As the ore was weighed after sampling, it seemed best to take the moisture sample at the same time. The moisture sample was quartered down and the final cut was put at once into hermetically sealed glass bottles. The moisture content was determined in the laboratory of the Bureau of Mines.

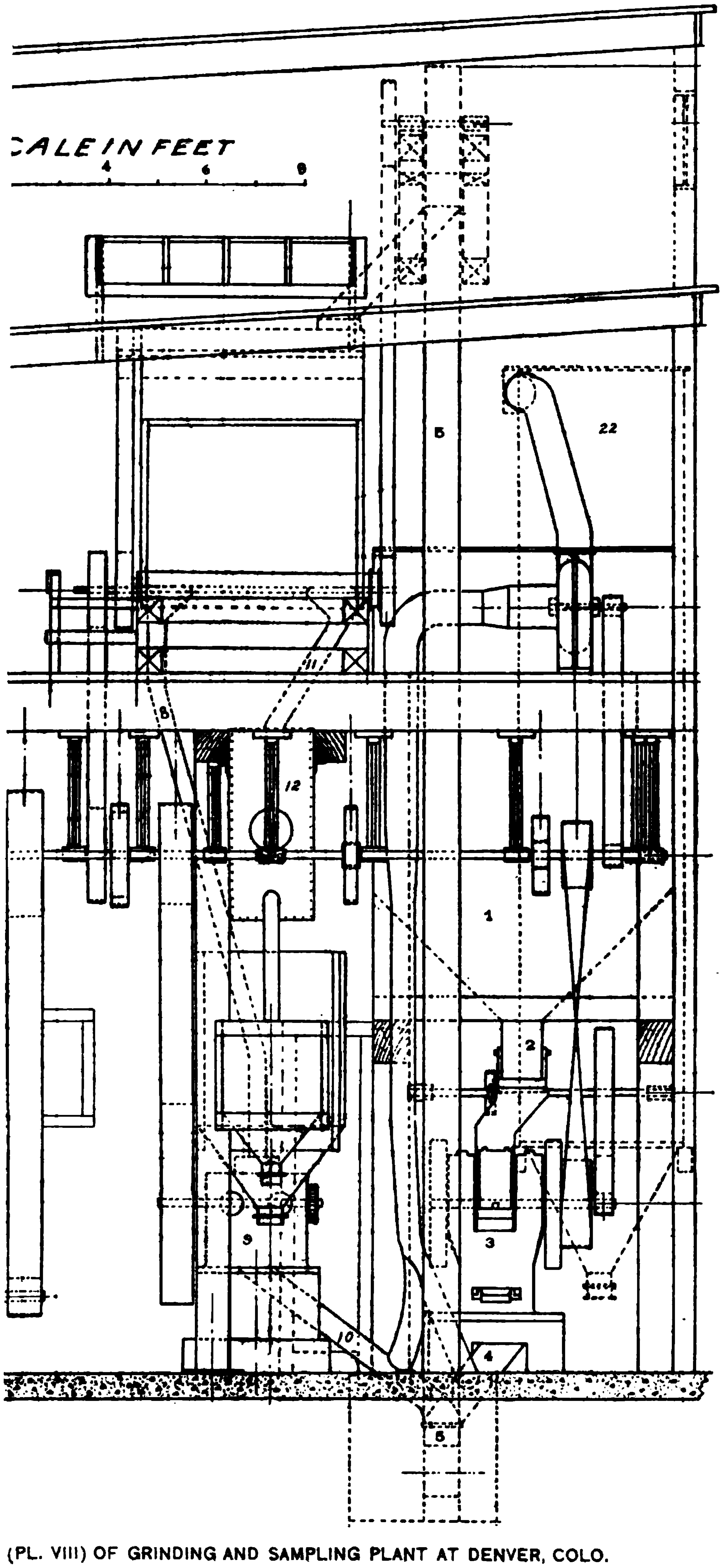


SECTIO

BUREAU OF MINES



SECTION *B-B*



(PL. VIII) OF GRINDING AND SAMPLING PLANT AT DENVER, COLO.

CHAPTER III. CONCENTRATION OF MILLING ORE.

In mining carnotite ore of shipping grade considerable low-grade ore is produced which is not rich enough in uranium content to warrant shipment, but may contain as much as 1½ per cent of uranium oxide, the average at the claims of the National Radium Institute being, according to observations of the authors, about 0.8 per cent. At many mines such ore formerly was not kept separate from the waste rock but was sent to the dump. The ratio between the amount of milling ore and the amount of shipping ore produced in mining varies greatly, as low as one-quarter ton of milling ore per ton of shipping ore being produced in places where the pockets were exceptionally rich, but in mining poor ore as much as 10 tons of milling ore per ton of shipping ore have been mined.

Early experiments showed, as stated in a previous publication,[a] that it was possible to concentrate the carnotite from these low-grade ores by either the wet or the dry method and thus prevent the relatively large waste of radium which the mining of carnotite had entailed. In most districts of the carnotite belt dry concentration is perhaps preferable, as water is scarce. Concentrating the low-grade ores reduces the bulk and increases the uranium content, thus effecting a considerable saving in haulage and freight and making shipment profitable. Where enough water is available wet concentration can be applied. As all the claims operated by the National Radium Institute were in an arid region, where no water for concentration was available, the dry method was chosen.

To investigate this problem and, if possible, to put the concentration of carnotite on a commercial basis, an experimental mill was erected by the National Radium Institute on the Maggie C claim, and extensive tests were conducted with low-grade ore. All milling ore mined on the various claims was carefully separated and placed in as available localities as possible. The ore on the Maggie C claim was used for the early experimental tests, about 50 tons of concentrates being produced, which contained 3 to 7 per cent uranium oxide. As these tests were a success the mill was enlarged and improved. Its equipment and the results obtained are described in a later chapter. Views of this mill are shown in Plates V, *B* (p. 40) and VI, *B* (p. 48).

[a] Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium: Bull. 70, Bureau of Mines, 1914, pp. 35–40, 111–112.

METHODS OF CONCENTRATING CARNOTITE.

As carnotite partly forms the binder between the sand grains of the sandstone, it must be separated from these grains in order to concentrate it. Consequently the ore should be reduced to about the fineness of the sand grains. Care must be taken that as little as possible of the silica is ground to powder, as the concentration depends largely on the removal of fines or slimes from the coarser material. Some of the carnotite adheres tenaciously to the surface of the sand grains and can be liberated only by attrition or elutriation. In the especially devised attrition apparatus, described later, the ground ore is rubbed by means of rotating brushes pressing against the surface of a circular plate.

In order to concentrate the fines or slimes, which contain most of the carnotite, either the wet or dry method of concentration can be used. In the wet method advantage is taken of the difference in rate of settling of different sizes of the agitated material, followed by decantation and final settling of the slimes, which form the concentrate. Dry concentration suggests the use of air to blow or suck the fines from the coarser particles.

A complete description of the dry methods used in the concentration mill of the National Radium Institute are given in the following pages. A description of wet methods of treatment is also included as being of general interest, although not used in this work.

ATTRITION OF THE GROUND ORE BEFORE CONCENTRATION.

It was found that after the ore had been ground to about 80 mesh or finer all of the carnotite could not be entirely liberated by ordinary means. In order to save this carnotite a special attrition apparatus was constructed whereby the adhering carnotite was rubbed or scraped from the silica grains mechanically by passing the ground ore between a pair of disks, on one of which was a number of stiff wire brushes. The disks were placed one above the other in a horizontal position, the brushes being attached in spiral form to the lower side of the upper disk. The lower disk was rotated by mechanical means. The material to be treated was fed through a hopper near the center of the rotating lower disk and was carried gradually toward the outer edge, the constant attrition by the wire bristles removing much of the adhering carnotite from the silica grains. The entire apparatus was made practically dust proof to save the liberated dust, which contained the valuable carnotite. The air current caused by the rotation of the disk tended to whirl up the liberated dust, which was removed through a hole in the upper side of the housing by a vacuum cleaner, and formed the concentrate.

The disks were 3 feet in diameter, and approximately 2 tons of material could be treated during 8 hours.

This capacity was insufficient for milling on a commercial scale, and the Raymond pulverizing and dust-collecting system was finally adopted. The manufacturers of that system had already made somewhat similar experiments with finely ground ore for other parties. This dry concentration, with previous coarse grinding, was ultimately chosen by the National Radium Institute for its ore and is described in detail later.

WET CONCENTRATION OF CARNOTITE ORES.

In the application of the wet method, the ore should be carefully ground to about 80 or 100 mesh. Before separation it will be necessary to elutriate the ground material in order to liberate the more tenaciously adhering carnotite from the silica grains. The elutriated material should then be agitated in tanks with ample water. After agitation the coarser particles are allowed to settle and the liquor, with the pulverulent carnotite and other fine material (slimes) which remain in suspension, is decanted into settling tanks. Water is added to the residue, and the operation repeated, as some of the carnotite is carried down by mechanical action with the coarser silica grains. A number of settling tanks should be employed, as it takes some time for all the slime to settle. After thorough settling of the slime the water can be drawn off and, if necessary, can be used again. The settled slimes form the concentrate.

Fischer [a] suggests that the crushed and ground ore be brought into a Dorr classifier, which mechanically separates the coarse material from the slimes, and thereby accomplishes concentration at the same time. The concentrates are carried by a launder into a Dorr thickener, where most of the water is eliminated. The moist material is then dried. The dried cake can be broken and put into bags. Fischer claims that a concentration of 3:1 can be effected in this way, the concentrates being three times as rich as the original low-grade ore. Therefore an ore containing 0.75 per cent U_3O_8, according to Fischer's figures, would give a concentrate containing only about 2.25 per cent U_3O_8. Such a product seems very low grade, and no doubt better results than these can be obtained by wet concentration.

J. V. N. Dorr suggested to the authors that in addition to the classifier and slime thickener, a number of settling tanks similar to those used in countercurrent decantation be provided in order to make the process more continuous. Such a method will require the use of elutriation in order to make a better recovery. The carnotite

[a] Fischer, Siegfried, The carnotite industry: Trans. Am. Electrochem. Soc., vol. 24, 1913, pp. 365-368.

adhering to the sand grains must first be liberated and agitation with water alone will not free it, as many tests have shown.

Although the Dorr classifier apparently ought to give good results in the wet concentration of carnotite ores, other systems of agitation followed by decantation and settling should also be of commercial use wherever enough water is available.

Nothing is known as to the cost of such methods, but it should be low.

PROCESS USED BY NATIONAL RADIUM INSTITUTE.

In concentrating low-grade ore the National Radium Institute, as previously stated, employed the dry method because the water supply at Long Park was sufficient for domestic use only.

DESCRIPTION OF PLANT DURING EXPERIMENTAL STAGE.

The concentration mill, built on the Maggie C claim, was equipped during the experimental stage with an ore bin, a 5 by 9 inch Sampson crusher, and a Raymond No. 00 pulverizing machine with a tubular dust collector. The crushed ore was dried before it was pulverized. During the experimental work, the drier was a steel plate one-fourth inch thick, 6 feet long and 3 feet wide, with edges turned up on three sides, which was placed on a low stone firebox, in which a coal or wood fire was built. The machines were driven by a 25-horsepower Fairbanks-Morse kerosene engine.

The Raymond machine had a beater chamber containing two sets of rapidly revolving beaters. Over the beater chamber was a cone of galvanized iron, within which was an inner cone provided at its lower end with a swinging discharge gate. The two cones were connected at their upper ends by a number of small gate shutters and had a common top or cover plate. A large pipe from the center of their top led to an exhaust fan mounted on the same shaft as the beaters. This fan discharged into a cyclone dust and air separator, which was connected by a return pipe with the beater chamber of the pulverizer. A small pipe led from this return pipe to a tubular dust collector. Both the cyclone separator and the tubular dust collector had discharge gates at their lower end.

The Raymond mill is used extensively to pulverize cement, paints, etc. In such work the beaters reduce the material to a fine powder. In the concentration of carnotite, however, such pulverizing proved unnecessary and even detrimental, the function of the beaters being chiefly to stir up the ground ore fed into the machine so that the carnotite dust could be removed from the larger grains of silica. With this machine the use of any other attrition was unnecessary, as the silica grains whirled around in the beater chamber rubbed against each other

and the lining of the chamber. The beaters also caused further disintegration of the material into grains, thus liberating the adhering carnotite.

The finer material was drawn by suction into the space between the outer and inner cones, but the coarse particles dropped out of the air current and fell back into the beater chamber. Here the stronger air current practically air washed the coarser particles free from the dust, after which they automatically dropped through the tailing spout at the front of the pulverizer. The finer material passed through the gate shutters into the inner cone, where a further separation took place, the finer particles being drawn through the fan, while the coarser particles fell to the bottom of the inner cone and were automatically discharged again into the beater chamber.

The volume of air used was large, and the air current was light enough to carry off the fine powder. The fan blew the dust into the cyclone separator, where the dust was separated from the air and automatically discharged as concentrate No. 1. This cyclone air separator furnished the bulk of the carnotite concentrate. The surplus air was returned through a connecting pipe from the top of the separator into the pulverizer; a certain amount of this surplus air blown into the tubular dust collector through a smaller pipe, furnished concentrate No. 2, which consisted of the very finest dust, and hence was richer than concentrate No. 1. However, the amount collected was comparatively small, only about 1½ per cent of the weight of concentrate No. 1.

The ore was dumped onto the drying plate in about 250-pound lots and stirred by hand with an iron rake until dry, much dust being carried away mechanically with the steam. This drier had a capacity of 4½ tons of ore a day with three shifts. The dried ore was fed by hand into the feed hopper of the pulverizer. During these experiments about 50 tons of concentrate No. 1, containing about 3.25 per cent uranium oxide, and 1,600 pounds of concentrate No. 2, with a uranium oxide content of about 6 per cent, were produced. The ratio of concentration was about 6:1. The recovery of U_3O_8 in the concentrate during the experimental stage was about 50 per cent.

The experimental mill required the following help: One foreman, one crusher man, one laborer to carry ore to the drier, three drier men (one for each of three shifts per day), and one man to feed the pulverizer and remove the tailings.

Coal and wood were used for the drier. The coal was hauled from a mine near Nucla, Colo., a distance of about 20 miles. The wood was local pinon and scrub cedar. The kerosene for the engine had to be hauled from Placerville, Colo., a distance of about 58 miles.

DESCRIPTION OF COMPLETED CONCENTRATION MILL.

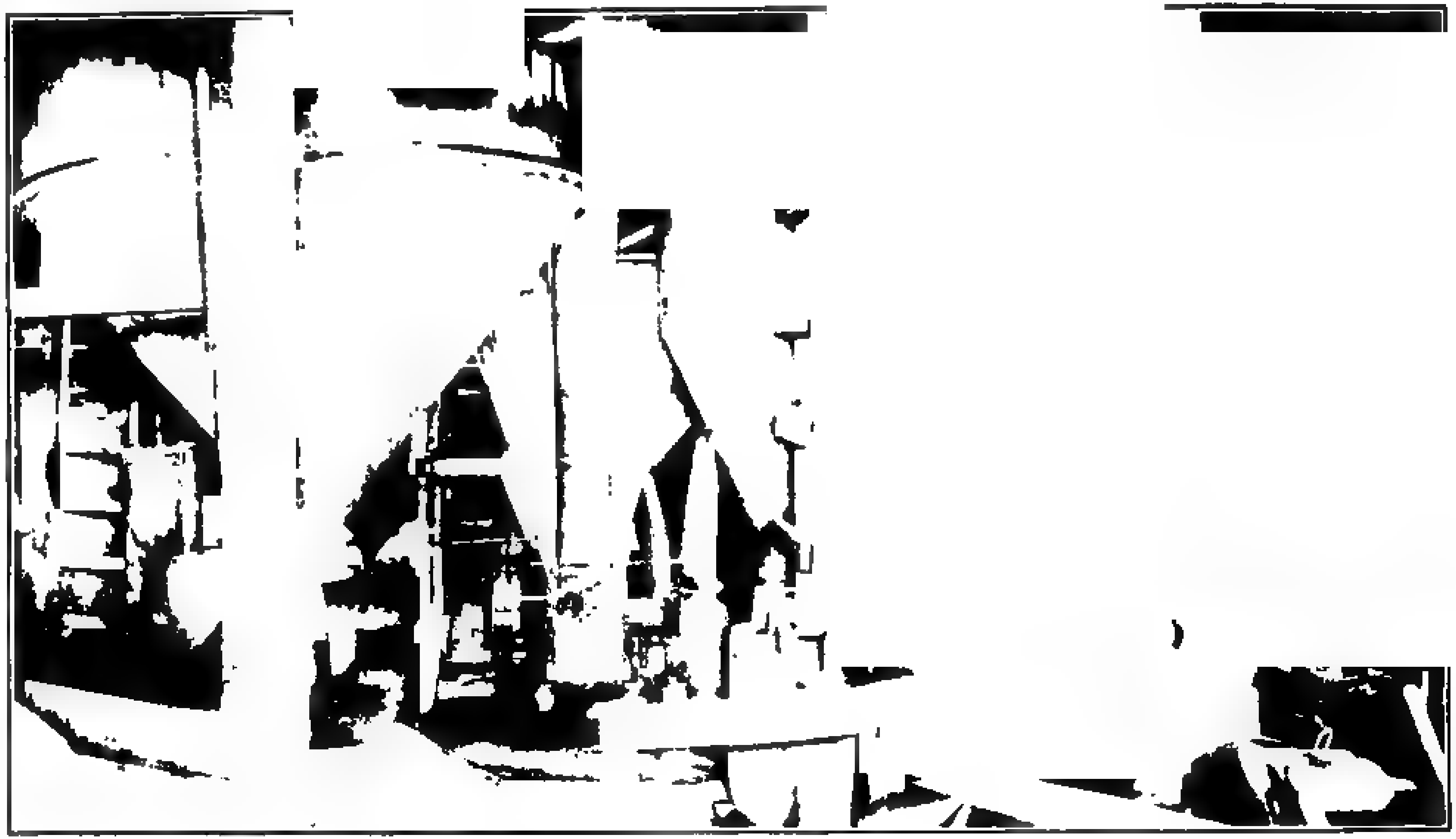
As the experimental results were satisfactory, the National Radium Institute decided to complete the mill and operate it on a commercial basis. In addition to the apparatus already described, the following equipment was installed: A mechanical drier and fan, a cyclone dust collector, 2 sets of rolls, 2 bucket elevators, an ore-belt conveyor, a tailings belt conveyor, a vibrating screen, additional fans, piping and dust collectors, and another 25-horsepower Fairbanks-Morse kerosene engine. Plate XI shows the interior of the mill.

The new equipment necessitated changes and additions to the mill building. The completed building was 38 feet 6 inches long and 30 feet wide, the greatest height was 23 feet; the engine room was 13 feet high on one side, sloping to 11 feet. On the north side, a small shed 10 by 10 by 10 feet, formerly used as the drying room, served as a storage room. All new additions had a light framework covered with painted corrugated iron.

Plates XII, XIII, and XIV show the plan view, section A–A, and section B–B, respectively. In the illustrations *1* designates an ore bin built on a foundation of piled rock. The bin was made of heavy timber and planks, fastened together with iron rods threaded at either end. It was 8 by 10 by 10 feet in size, and held about 40 tons of milling ore. At its lower end the bin had an opening about 8 inches square, beneath which was a swing-gate feeder *2* that discharged into a chute, *3*, placed at an angle of 45°. The upper part of the bottom side of this chute held a strong wire screen with ½-inch apertures. The chute ended in a spout that discharged through a pipe into the boot *5* of elevator No. 1. The screen of the chute extended to the feed hooper of a 5 by 9 inch Sampson crusher, *4*. The discharge spout of the crusher connected with elevator boot *5*.

This elevator had 3½ by 5 inch steel buckets mounted on a 6-inch rubber belt, and was housed in 8-inch pipes. Belt conveyor *6* had a 10-inch rubber belt supported on wooden rollers. The two pulleys were mounted on a framework of 4 by 4 inch lumber. Underneath the conveyor frame was a semicircular pan to keep pieces of rock from falling into the mill below.

At the discharge end of this conveyor a small hopper and pipe, *7*, connected with the feed hopper *8* of a Ruggles-Cole single-shell rotary drier, *9*, with a shell 12 feet long and 3 feet in diameter. The smoke-stack, *22*, at the upper end of the drier had in one side an opening connected by a pipe with a Buffalo No. 5 blower, *23*. Both the smokestack and the pipe had dampers. The smokestack was used when the fire in the drier was started. The blower, which was used exclusively during operation, was connected through another pipe with a cyclone dust collector *24* on the roof of the building.



INTERIOR VIEW OF CONCENTRATING MILL AT LONG PARK, COLO

The outlet at the top of the cyclone was 28 inches in diameter, and was connected through a 28-inch pipe with an especially constructed dust collector, or bag house *24a*, consisting of a cylindrical upper compartment or tank made of No. 16 galvanized iron, and a lower compartment which was cone shaped. These were 7 feet 6 inches in diameter, and were connected by an inner hose made of burlap, which was surrounded by another hose made of unbleached muslin, the length of each hose being 10 feet. The upper compartment had a circular opening at the top 3 feet in diameter, over which was stretched a piece of burlap. The gases escaped partly through the hose and partly through the burlap; the dust was collected inside and was drawn off at the opening in the lower end of the cone.

Trial runs had shown that much dust escaped from various points in the apparatus. In order to save this dust, a system of 3-inch galvanized iron pipes was attached at the points where the chutes from the crusher and screen and those from the drier and rolls entered the upper side of the boots of elevators Nos. 1 and 2, respectively. These pipes, which were almost vertical, connected with a 7½-inch pipe leading to the intake of a Buffalo No. 3 blower fan. Another pipe 7½ inches in diameter was connected with the outlet of this blower, and at its lower end was fitted into the top of dust collector *24*. The blower had a dust trap in its intake and another trap in its outlet. At the lower end of the drier and near the fire box *9a* a discharge spout, *10*, led into the boot, *11*, of the elevator No. 2. This elevator had 3½ by 5 inch steel buckets fastened to a sprocket chain and was housed completely, 8-inch pipes being used between the boot and the head. Its discharge spout was soldered to the feed hopper of a vibrating screen, *12*. A small screw conveyor in the upper part of the screen housing fed the ore to a ¼-inch screen, 4 inches below which was a 10-mesh screen. Both screens were placed at an angle of 45°. The entire screen housing was made practically dust proof, and two spouts at the lower end ended in 4-inch pipes of No. 16 galvanized iron which connected with the feed hopper of a set of 9 by 12 inch McFarland rolls, *13*, and a set of 12 by 12 inch Sturtevant rolls. The latter are not shown in the plates, as they were installed later. The oversize of the ¼-inch screen went to the McFarland rolls and the oversize of the 10-inch screen to the Sturtevant rolls for further grinding. The material passing through the 10-mesh screen went to the storage hopper, *14*, of the pulverizer. Discharge spouts from the rolls led into the boot, *11*, of elevator No. 2. Nos. *15* to *19* designate the Raymond mill and the dust-collecting system described. Below the tailings spout of the pulverizer, *16*, was installed a horizontal tailing belt conveyor, *25*, 8 feet long, which discharged at right angles on a tailing belt conveyor *25a*, 35 feet long, with a 17 per cent upgrade, which extended

through the east side of the building, the outer end being supported on a scaffold about 25 feet from the building. This conveyor was of rubber belting 10 inches wide supported by rollers, with a 12-inch iron pulley at each end.

Four water tanks, *27*, outside the building furnished the cooling water for the two 25-horsepower engines, *26*. Kerosene feed tanks, *28*, were buried in the ground near the water tanks. Power was transmitted by two shafts; one making 530 revolutions per minute drove the Raymond machine and the tailings conveyors. The other shaft, making 200 revolutions per minute, drove the rest of the machinery. Thus the mill could have been operated in two

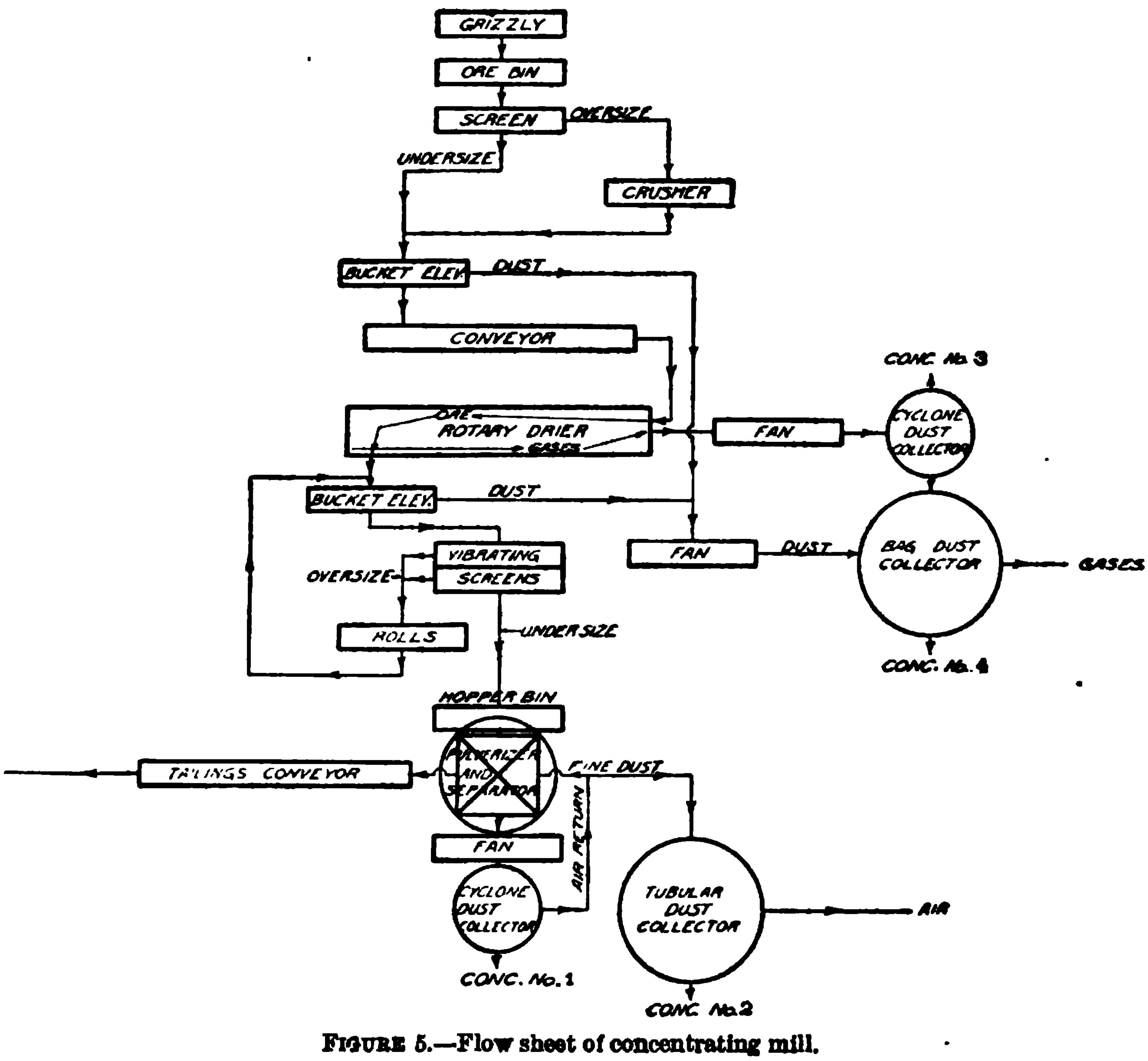


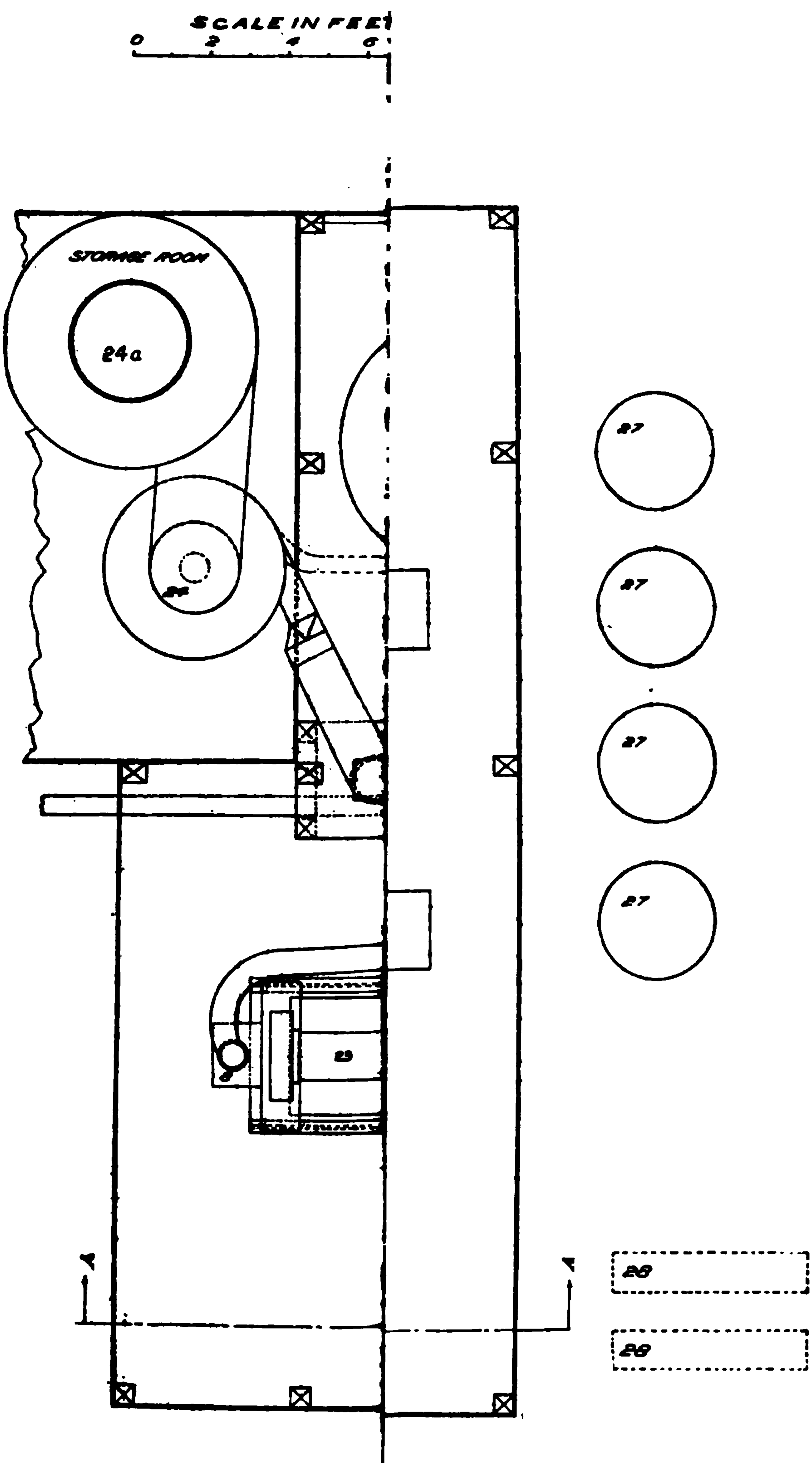
FIGURE 5.—Flow sheet of concentrating mill.

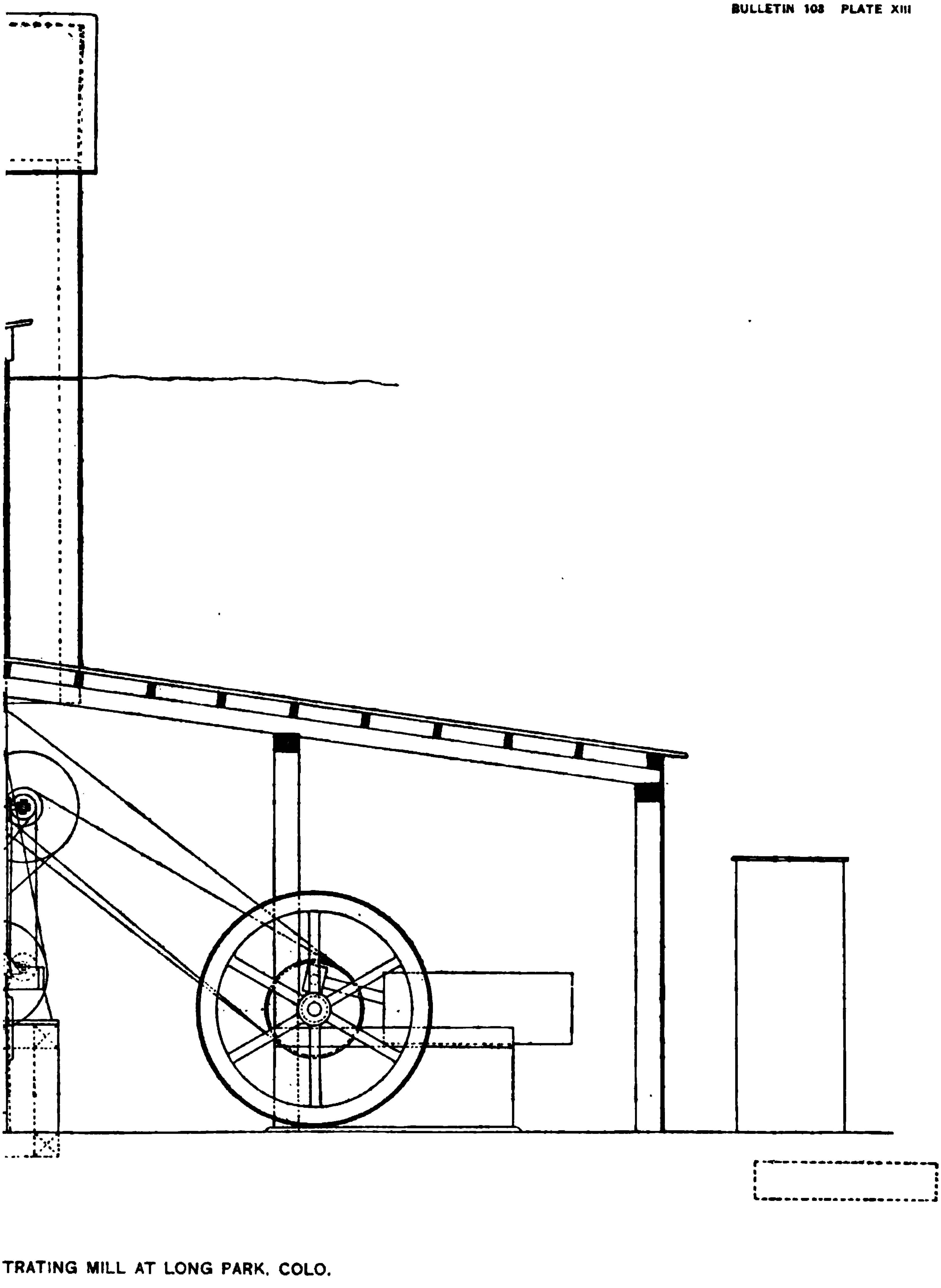
units if necessary on account of a breakdown of either unit. Ample provision was made for storage of the ground ore in the pulverizer hopper, *14*.

A coal storage bin was built at the east side of the building, and a smaller one inside for daily use.

COURSE OF ORE IN THE MILL.

The course of the ore through the mill is indicated by the flow sheet shown in figure 5. The milling ore was dumped onto the grizzly (Pl. V, *C*, p. 40), which had bars spaced 1½ inches apart; the larger pieces


SCALE IN FEET
0
2
4
6
STORAGE ROOM
24a
27
27
27
27
A
A
28
28



TRATING MILL AT LONG PARK, COLO.

of rock were broken with a rock hammer. Pieces of waste rock were thrown out. From the ore bin the ore was fed mechanically onto a ½-inch screen placed at an angle of 45°. The oversize dropped into the crusher, the jaws of which were set to make a ½-inch product. The undersize dropped through a chute into the boot of elevator No. 1, as did the material passing through the crusher. The material discharged from the elevator was carried by belt conveyor No. 1 across the building and dropped through a pipe into the feed hopper of the rotary drier. In passing through the drier the ore was repeatedly lifted by the shelves within and poured through the hot gases coming from the fire box at the lower end. Considerable dust was liberated, which was carried with the gases and steam into the smokestack. To save this dust all gases and steam were drawn off by the fan through the pipe connected with the smokestack and blown into the cyclone dust collector. Through a pipe at the lower end of this collector the separated dust was drawn off. This dust formed concentrate No. 3.

Considerable very fine dust carrying carnotite escaped through the opening at the top of the cyclone separator and was blown through another pipe into the dust collector, or bag house. The dust so collected formed concentrate No. 4. The dried ore left the drier at its lower end near the fire box and discharged into the boot of elevator No. 2. The hot ore was elevated to the vibrating screen, where the screw conveyor distributed it equally over the screening surface. The oversize of both the ¼-inch and 10-mesh screens fell through pipes into the feed hoppers of the two sets of rolls; the undersize passed through a chute to the storage hopper of the Raymond mill. The screened material consisted of ground ore as shown by the following screen test:

Results of screen test of ground ore as fed to Raymond mill.

	Per cent.
On 35 mesh	13.8
On 48 mesh	18.0
On 65 mesh	27.4
On 100 mesh	19.6
On 150 mesh	13.2
On 200 mesh	2.7
Through 200 mesh	5.3
	100.0

The oversize, after passing the rolls, dropped again into the boot of elevator No. 2, and was returned to the screen until all of it passed through the 10-mesh screen. The ground and dried ore from the storage hopper was automatically fed into the pulverizer chamber of the Raymond mill, when the carnotite was separated from the silica grains, as already described. The tailings, which automatically dis-

charged from the pulverizer, dropped onto the tailing conveyor and were carried to the dump. The size of the tailings corresponds to the following screen test:

Results of screen test of tailings.

	Per cent.
On 35 mesh	3.0
On 48 mesh	8.0
On 65 mesh	20.4
On 150 mesh	57.0
On 200 mesh	11.6

The tailings were practically free from carnotite dust.

The blower fan connected with the boots of elevators No. 1 and No. 2 drew off the dust from these points and blew it into the dust collector already described.

It is essential in concentrating carnotite ore by the dry method that all apparatus be so inclosed and connected with each other as to make the entire unit dust proof, for all dust created in the mill contains carnotite and is part of the concentrate.

As already stated the concentrates were of four kinds, as follows: Concentrate No. 1, from the cyclone separator of the Raymond mill; concentrate No. 2, from the tubular dust collector; concentrate No. 3, from the cyclone separator for the mechanical drier and from the two dust traps connected with the auxiliary dust-collecting fan; and concentrate No. 4, from the dust collector or bag house receiving the gases, dust, and steam from the drier cyclone. The intake of the last was also connected through the blower fan with the crusher, elevator No. 1, the drier discharge spout, the rolls, and elevator No. 2, and all dust drawn off from them went into concentrate No. 4. The relative percentages of the four concentrates obtained in operating the mill were approximately as follows: No. 1 concentrate, 88 per cent; No. 2 concentrate, 2 per cent; No. 3 concentrate, 7 per cent; and No. 4 concentrate, 3 per cent.

In order to sample the feed and the tailings operation and estimate the approximate weight of ore fed or tailings discharged, small sampling devices were installed.

FEED SAMPLER.

The feed sampler was a chute or trough that passed periodically under the stream of ore from the automatic discharge hopper on the bottom of the ore bin, and thus removed a small sample.

This chute was made of No. 16 gage sheet iron and was 4 inches square and 3 feet long. It was pivoted at its lower end, the upper end being supported by a piece of strap iron, and was set at an angle of 40° with the horizontal. The movement of the sampler was pro-

duced through a rod and lever actuated by a trip on a 3-inch belt. This belt was 17 feet 6 inches long and made one complete revolution every 50 seconds, pushing the chute under the ore stream once every revolution. A counterweight returned the chute to its original position. A sack at the lower end of the chute caught the portions of ore.

The sample obtained, which was something less than a hundredth part of the weight of the feed, was quartered by hand to approximately 25 pounds; then broken by hand to about ¼-inch mesh; and then reduced to 1 pound through a split sampler. The residue after being ground on a bucking board to pass a 40-mesh screen was reduced by a split sampler to approximately a quarter of a pound. This sample was tested with the electroscope.

TAILINGS SAMPLER.

For the tailings sampler more room was available, so that a sampler of somewhat different type was evolved. A sheet-iron tray 11 inches long, 1 inch deep, and 1⅝ inches wide, with sides carefully filed to a knife edge, was suspended from ¼-inch rod hung between two parallel endless sprocket chains placed 14 inches apart. These chains were of No. 25 link belt and were 162½ inches long, or exactly a hundred times the width of the tray. The sample represented just 1 per cent of the weight of the tailings.

The chains were driven by 3-inch sprockets. In passing the sprocket the tray turned completely over and its contents fell into a sack. The chain, guided by idler pulleys, carried the tray so that it passed through the stream of sand falling from the pulverizer discharge onto the conveyor and took one complete cut of this entire stream every 35 seconds. At the end of each day's run this sample was carefully weighed, and by multiplying this result by 100 the total weight of tailings produced was determined. The sample was usually fine enough to pass a 20-mesh screen and was reduced in a split sampler to the amount required for electroscopic tests.

SAMPLING OF CONCENTRATES.

All four products, concentrates Nos. 1, 2, 3, and 4, were sampled by means of a pipe sampler.

As carnotite concentrates are a very fine dust or powder which, being hygroscopic, quickly absorbs moisture from the air, they pack tightly, so that mechanical sampling is impracticable. It was found most suitable to sample the concentrates by means of a pipe or "cheese scoop" sampler, made of a piece of brass pipe about one-half inch in diameter and 30 inches long. About one-third of the pipe is removed to form a slot; that is, a strip lengthwise of the pipe is cut out, with the exception of about 4 inches on one end, to which a wooden handle is fastened. The lower end of the pipe is rounded

off and slightly sharpened. This sampler was pushed into each bag of concentrate in two different places. The pipe, filled with a representative cut of the concentrate, was then drawn from the bag and its contents were pushed out or the pipe was lightly tapped with a piece of wood. Two samples were taken from each bag. Each bag should be sampled in the same places in order to insure the sample being representative. The gross sample obtained was thoroughly mixed and coned, quartered, and finally split-sampled to the desired quantity.

ELECTROSCOPIC TESTS OF CONCENTRATES.

Field tests of concentrates in the electroscope are in general the same as for ore, as described under "Testing Ores." An important fact to be remembered is that the heat required to dry the ore for the Raymond mill drives off a certain amount of radium emanation from the ore, the amount varying with the degree and duration of heating. One-half of the emanation thus driven off will accumulate again in a period of 3.9 days, although approximately 30 days are required for the emanation to regain full strength. The amount of emanation driven off in heating the ore in the mill of the National Radium Institute during the winter months was approximately 5 per cent. Therefore, as this loss of emanation affected the apparent result in the electroscope, if the radioactive strength of the fresh concentrates showed a uranium oxide content of 3 per cent, it was safe to assume that they actually contained about $1.05 \times 3 = 3.15$ per cent. If, however, the test was not made until four days after milling, the error was only 2½ per cent, which could be neglected because of its being less than the limit of accuracy with the electroscopic method. As proper mill control requires frequent tests soon after the concentrates are produced, a correction to be determined from time to time must be added to the result obtained by the electroscopic test.

TABULATED DATA ON OPERATION OF MILL.

The accompanying table gives complete data on the dry concentration of milling ore at Long Park, Colo. The results were obtained from operations on a commercial scale during the first months' run with the complete equipment and give an accurate idea of what can be expected from the dry process employed. The capacity of this mill was a little over 1 ton of milling ore an hour and the average output was 365 pounds of combined concentrates (Nos. 1, 2, 3, and 4) an hour.

The milling ore had an average content of 0.85 per cent U_3O_8; the average U_3O_8 content of the four kinds of concentrates was as follows: No. 1, 2.91 per cent; No. 2, 3.55 per cent; No. 3, 2.76 per cent; No. 4,

3.44 per cent; the total average of all four concentrates combined was 2.92 per cent U_3O_8.

Of the quantity of concentrates produced 87.77 per cent were collected as No. 1, 1.75 per cent as No. 2, 7.39 per cent as No. 3, and 3.09 per cent as No. 4. The tailings averaged 0.37 per cent U_3O_8.

The relatively high content of U_3O_8 in the tailings can not be conveniently recovered therefrom by only one mechanical operation, as microscopic examination shows that the carnotite left in the tailings is partly in pure hard particles about the size of the silica grains. Such particles could not be disintegrated without grinding up a large amount of the silica, which would go into the concentrates, reducing their uranium oxide content; also a small amount of carnotite is inclosed between the fractures of silica grains as fine as 150-mesh, and can not be liberated without grinding these grains to fine dust. However, the tailings are practically free from all adhering dust and carnotite.

The ratio of concentration was about 6:1, about 6 tons of milling ore yielding 1 ton of concentrates. Feeding richer ore, of course, lowers this ratio.

As much as 63.7 per cent of the carnotite contained in the milling ore was extracted and 60 per cent of all the carnotite in this ore was actually recovered in the concentrates, on the basis of the uranium content obtained by chemical analysis. In this statement "extraction" is considered as the difference in uranium oxide content of the feed and the tailings, whereas "recovery" is considered as the actual uranium content of the concentrate expressed as a percentage of the uranium content of the feed.

SUPPLIES USED.

In the operation of the mill, as shown by the table following, there were used 15.45 gallons of kerosene, 1.75 gallons of gasoline (to start engines), and about 283 pounds of coal per ton of concentrates produced; 2.7 gallons of kerosene and approximately 50 pounds of coal for each ton of ore milled were necessary to reduce the moisture in the ore to less than 1 per cent. The coal used was the best available cheaply in that section of the country; it has a heating value of about 10,000 British thermal units per pound.

During the period covered by the data presented the mill operated only one shift daily. Since then the mill has run with two shifts daily, but the figures obtained during this work were practically the same, with the exception of a slight saving on labor costs, as those given in the table following.

Results of first month's operations of complete concentration mill at Long Park, Colo.

Run No.	Time, hours.	Feed, dry weight (pounds).	U_3O_8 content of feed.		Dry weight of concentrates, pounds.					U_3O_8 content in concentrates.								
										Per cent.				Pounds.				
			Per cent.	Pounds.	No. 1.	No. 2.	No. 3.	No. 4.	Total.	No. 1.	No. 2.	No. 3.	No. 4.	No. 1.	No. 2.	No. 3.	No. 4.	Total.
1	53½	111,100	0.8	888.8	16,680	300	1,630	382	18,992	2.9	3.5	2.7	3.21	483.7	10.5	44.1	12.3	550.6
2	61	123,000	.9	1,107.0	17,695	430	1,550	596	20,271	3.0	3.5	2.8	5.53	530.8	15.0	43.4	21.0	610.2
3	37	81,000	.8	648.0	11,700	160	1,360	453	13,673	2.9	3.9	2.85	3.49	339.3	6.3	38.7	15.8	400.1
4	48	98,500	.9	886.5	17,820	205	1,200	535	19,760	2.84	3.45	2.7	3.5	506.0	7.1	32.4	18.7	564.2
Total	199½	413,600		3,530.3	63,895	1,095	5,740	1,966	72,696					1,859.8	38.9	158.6	67.8	2,125.1
Average			.85							2.91	3.55	2.76	3.44					

Run No.	Time, hours.	Feed, dry weight (pounds).	Tailings dry weight (pounds).	U_3O_8 in tailings.		Extraction, per cent.	Recovery, per cent.	Feed capacity, pounds.[a]	Amount of concentrates produced per hours' run, pounds.	Relative quantities of concentrates produced, per cent.				Ratio of concentration.
				Per cent.	Pounds.					No. 1.	No. 2.	No. 3.	No. 4.	
1	53½	111,100	92,100	0.33	304.0	67.03	63.18	2,086	356	87.85	1.58	8.53	2.01	5.85 : 1
2	61	123,000	102,700	.40	410.0	62.96	55.12	2,016	332	87.30	2.12	7.64	2.94	6.06 : 1
3	37	81,000	67,300	.35	235.5	63.65	61.74	2,189	369	85.57	1.18	9.94	3.31	5.92 : 1
4	48	98,500	78,500	.42	329.7	63.93	63.64	2,052	411	90.19	1.03	6.07	2.71	4.98 : 1
Total	199½	413,600	340,600		1,279.2									
Average				.375		63.7	60.2	2,086	365	87.77	1.75	7.39	3.09	5.69 : 1

SUPPLIES USED.

Run No.	Kerosene used.		Gasoline used, engines Nos. 1 and 2.	Coal used for drier.	Production of concentrates.	Kerosene used per ton of concentrate.	Gasoline used per ton of concentrate.	Coal used per ton of concentrate.	Quantity of milling ore treated.	Kerosene used per ton of milling ore.	Coal used per ton in drying of milling ore.
	Engine No. 1.	Engine No. 2.									
	Gal.	*Gal.*	*Gal.*	*Lbs.*	*Lbs.*	*Gal.*	*Gal.*	*Lbs.*	*Lbs.*	*Gal.*	*Lbs.*
1	72	92	14	2,650	18,992	17.27	1.49	279.00	111,100	2.95	47.70
2	86	91	16	3,350	20,271	17.46	1.57	330.52	123,000	2.87	54.47
3	43	53	11	2,060	13,673	14.04	2.34	301.32	81,000	2.37	50.86
4	56	73	16	2,200	19,760	13.05	1.61	222.66	98,500	2.61	44.67
Total	257	309	57	10,260	[b] 72,696	61.82	7.01	1,133.50	[c] 413,600	10.80	197.70
Average per ton of concentrates						15.45	1.75	283.38		2.70	49.42

[a] Dry weight, or 2,200 pounds wet weight.

[b] 36.35 tons.

[c] 206.8 tons.

PRODUCTION OF CONCENTRATES.

The following table shows the amount of concentrates produced at Long Park from the beginning of the experimental work to the closing of operations in August, 1916.

Production of concentrates to August, 1916.

[The net weights are for dry concentrates.]

Month.	No. 1 concentrate.		No. 2 concentrate.		No. 3 concentrate.		No. 4 concentrate.		Total.	
	Number of bags.	Net weight, pounds.	Number of bags.	Net weight, pounds.	Number of bags.	Net weight, pounds.	Number of bags.	Net weight, pounds.	Number of bags.	Net weight, pounds.
1915.										
(a)	2,119	103,620	86	1,660	....	....	....	....	2,205	105,280
November	283	25,496	46	3,663	53	4,744	....	....	382	33,903
December	155	13,950	39	3,510	37	3,330	....	....	231	20,790
1916.										
January	127	11,250	35	3,080	23	2,070	....	....	185	16,400
February	574	50,545	36	2,205	46	4,170	22	1,273	678	58,193
March	565	50,950	17	705	47	4,040	24	1,518	653	57,213
April	652	57,885	22	882	57	5,100	20	1,260	751	65,127
May	904	81,215	42	1,760	135	12,090	25	1,488	1,106	96,553
June	919	79,460	34	1,569	218	19,985	34	2,145	1,205	103,159
July, August	341	30,879	18	795	82	7,790	11	687	452	40,151
Total	6,639	505,250	375	19,829	698	63,319	136	8,371	7,848	596,769

a Produced during experimental tests.

RECOVERY OF CARNOTITE FROM LOW-GRADE ORES BY MECHANICAL CONCENTRATION.

Owing to the difficulty of separating the carnotite from the silica, extraction to be expected from mechanical concentration of low-grade ores is only relatively high. At least this is true of any of the methods that were fully tested. Therefore a recovery of 60 per cent of the carnotite in this low-grade ore, based on its known radium and uranium content, can be regarded as satisfactory with the equipment at hand.

Possibly with larger machines than those employed by the National Radium Institute a higher grade of concentrate can be obtained, as a large machine having a greater volume of air at lower velocity will separate much of the finely ground silica from the carnotite dust, but the concentrate would necessarily be considerably finer. Such machines, however, would be more costly, require more power, treat a larger tonnage—which is not always easily obtained from any one locality—and the extreme fineness of the concentrate would be probably a disadvantage in any acid leaching process for recovering radium.

POSSIBILITY OF RECOVERING CARNOTITE FROM MILL TAILINGS.

Tests were made to determine the possibility of a further recovery of carnotite from the mill tailings, which ordinarily contained about 40 per cent of the carnotite in the original feed. During one such test covering a day's run, in which about 5 tons of tailings were re-treated, a recovery of approximately 65 per cent was obtained, thus making the total recovery from the original mill feed approximately 85 per cent.

The National Radium Institute had only one Raymond machine in its mill, and, as operations had to be discontinued on account of the expiration of contracts, the tailings were left on the dump. Tests made during this work, however, show that in a new plant it may be advantageous to install a second Raymond mill to treat the tailings from the first. With one Raymond mill the material has to be rushed through in order to treat enough ore to make operation profitable. Under such conditions all of the carnotite can not be disintegrated, as the work of the beaters is largely to stir up the ground ore so the dust can be removed, but by running the dust-free tailings through a second Raymond machine the beaters can then act chiefly as grinders to liberate adhering carnotite and disintegrate grains that are not ground fine enough during the first operation. In re-treating tailings, the percentage of U_3O_8 in the concentrate can not be made as large as in the first operation, because silica is also necessarily ground up and is carried over into the concentrates, but a product averaging 1¾ per cent can be expected. However, running the additional machine adds comparatively little to the cost of operation and requires no additional labor, so that such re-treatment no doubt can be done at a profit.

Chapter IV. COSTS OF PRODUCTION.

The figures and costs given in the following pages include all expense for labor, supplies, repairs, insurance, general expense, and proportional amounts for amortization of equipment. The figures also include all costs of experimental work and equipment, as well as amounts credited to the Crucible Steel Co. for royalty on the ore mined from its claims, and the cost of cooperation of the Bureau of Mines. The cost of the shipping ore is given in a round sum, and represents an average of the costs over the entire period of mining by the National Radium Institute ended October 15, 1915.

The cost of concentrating the milling ore is also the average of the costs of concentrating during a period of several months.

SHIPPING ORE.

In calculating the cost of the shipping ore, containing about 2.6 per cent uranium oxide, or 52 pounds per ton of ore—three different figures are given in order to show the cost of the ore at the mine, at Placerville, Colo., and delivered at the sampling works of the National Radium Institute plant at Denver, Colo. These costs were as follows: Actual cost of mining this ore, which includes sorting but not sacking, approximately $36 per ton; cost of the ore at Placerville, approximately $61 per ton; and cost delivered at the Denver plant of the National Radium Institute, including all overhead charges and royalty, freight, and other expenses, $91.

These figures represent the average cost for approximately 960 tons of shipping-grade ore mined during about 15 months, and include 70 per cent amortization on mine equipment and buildings. Ordinarily this depreciation would be distributed over a longer period.

If cost figures for the various phases of the mining work, such as prospecting, sorting, and removing waste, are desired, the reader is referred to the chapter on mining, where various tables show the approximate percentage of time spent on each item; from these data and the values in the table following, it will be easy to compute such costs.

As regards mining cost of shipping and of milling ore, whenever milling ore was so associated with shipping ore that the milling ore had to be mined in order to get the shipping ore, the charge for mining the shipping ore included the cost of removing the milling

ore, but when the milling ore was mined by itself, the cost of mining it was separately entered and was not charged against the shipping ore.

Expenditures made by the mines department of the National Radium Institute during period of operation ended October 15, 1915.

Mine equipment and construction of mine warehouse	$980.49
Cost of automobile and live stock	655.00
Total cost of mining equipment	1,635.49
Cost of labor, net wages miners and foreman	20,550.00
Cost of commissary	11,481.75
Cost of mine supplies (including tools, steel, powder, fuse, caps, candles, water bags, blacksmith coal, etc.)	2,017.36
Cost of feed for live stock at mine	463.50
Total chargeable directly to mine operation	34,512.61
Cost of ore bags and twine	4,365.00
Cost of hauling ore to Placerville	19,913.63
Total cost for delivering ore at Placerville	24,278.63
Cost of loading ore into cars at Placerville and freight to Denver	8,245.00
General expenses at the mine (including office expenses, salary of office assistant at mine, telephone, telegrams, etc.)	1,516.98
Cost of maintenance of automobile	500.00
Cost of cooperation with the United States Bureau of Mines (including salary and travel expense of all members connected with the mining work).	6,906.90
Liability insurance, miner's compensation law	597.85
Total cost of general and overhead expenses	9,521.73
Cost of amortization on equipment (70 per cent of total cost, $1,635.49)	1,144.84
Cost of royalty to company owning claims	10,052.41
Total cost of royalty and amortization	11,197.25

SUMMARY.

		Total.	Per ton.[a]
Cost of mining ore	$34,512.61		
Cost of ore at the mine		$34,512.61	$35.96
Cost of delivering at Placerville	24,278.63		
Cost of ore at Placerville		58,791.24	61.24
Cost of loading and freight	8,245.00		
Cost of general and overhead expense	9,521.73		
Cost of royalty and amortization	11,197.25		
Total cost of ore delivered at Denver, including all charges		87,755.22	91.41

[a] Based upon production of 960 tons of shipping grade carnotite ore, net wet weight.

Commissary account.

Amount deducted for board from miners' wages	$11,075.95
Amount collected for board by petty cash account	244.08
	11,320.03
Commissary supplies on hand Oct. 31, 1915	500.00
	11,820.03
Amount paid for provisions and hauling to camp and cook's wages	11,481.75
Amount applied to amortization on camp equipment costing $876.90	338.28

As the National Radium Institute had contracted to do the assessment work on the 10 leased claims of the Crucible Steel Co. at Long Park, three years' assessment work, amounting to over $3,000, is included in the cost of mining the ore. Part of this, however, can be considered as actual cost of mining, as enough ore was obtained during assessment work to pay for such work. However, several claims were either nonproductive or produced so little that the assessment work, as far as the institute is concerned, must be considered a dead loss. The above calculation includes a charge of $10,052.51 for royalty, which of course would not have to be considered by a company working its own claim. However, such a company must charge off depreciation, and this charge should be considerably less than the royalty mentioned above. The actual cost of a pound of uranium oxide in the ore as mined by the National Radium Institute was $1.81 delivered at the Denver plant.

COST OF TRANSPORTATION OF ORE.

At first the National Radium Institute had its ore hauled to Placerville under contract at $20 per ton, all provisions and supplies, except machinery, being hauled to the mines free of charge. Later the contract was changed so that the rate was made $18 per ton and all haulage to the mines was paid for. As supplies were purchased at various points along the road, the prices for such back haul varied, as follows: From Placerville to the mines, $16 per ton; from Norwood, about 18 miles from Placerville, $11 per ton; from Redvale, $6.50 per ton; and from Naturita, the nearest town, $4 per ton. The mail and parcel post were carried free of charge.

As previously stated, most of the ore was loaded directly into the wagons, either at the claim or at the warehouse on the Maggie C claim. The ore from the outlying claims where the wagon could be driven up to the claim was loaded there. Some ore had to be packed a short distance to the road, as on the Great Western, Florence, and Bitter Creek claims.

For hauling the milling ore from outlying claims, and some of the milling ore mined underground on the Maggie C claim, to the ore bin of the mill a two-horse team, wagon, and driver were hired at $5 a

day. The teamsters staying over night at Long Park were boarded by the contractor, who had erected a small camp at a neighboring claim.

The freight rate on carnotite ore from Placerville to Denver was $8 per ton, when a valuation of $100 per ton was placed on the ore. A rate of $6 per ton could have been obtained by reducing the estimated value of the ore to $5 per ton, but the National Radium Institute found it of advantage to value the ore at $100 per ton.

CRUSHING, GRINDING, AND SAMPLING OF THE SHIPPING ORE.

Before the National Radium Institute installed its own grinding and sampling plant at Denver, it had to pay $4 to $6 per ton for the crushing, grinding, and sampling of its ores. The cost of grinding and sampling in its plant averaged $1.87 per ton of ore. This figure also includes 25 per cent amortization on the cost of building and equipping the plant.

Expenditures on grinding and sampling plant at Denver, Colo.

Equipment and construction		$3,177.64
Cost of operation:		
Labor	$726.30	
Supplies and repairs (including labor)	247.09	
Power	101.12	
	1,074.51	
25 per cent amortization on $3,177.64	794.41	
Total cost for grinding and sampling 1,000 tons of ore		1,868.92

ROYALTY.

The royalty credited to the company owning the claims for all of the ore mined from the claims to October 15, 1915, on approximately 931 tons, net dry weight, of carnotite ore, amounted to $10,052.41. The agreement was that the claims would be leased to the institute on a 15 per cent royalty basis, whereby 15 per cent of the ore mined from the claims was to be purchased by the institute at the price of $2 per pound of uranium oxide content, f. o. b. Denver, Colo.

Royalties credited to company owning the claims for ore mined to October 31, 1915.

Car No.	Amount.	Car No.	Amount.	Car No.	Amount.
1	$268.37	10	$331.08	19	$494.25
2	435.30	11	386.82	20	416.60
3	425.68	12	421.11	21	438.77
4	460.64	13	495.12	22	387.96
5	381.21	14	333.57	23	374.53
6	321.38	15	540.68	24	461.40
7	335.68	16	505.99	25	219.15
8	389.69	17	407.19		
9	367.32	18	452.92	Total	10,052.41

Fifteen per cent of ore containing 48,344 pounds U_3O_8=7,251.6 pounds U_3O_8, at $2=$14,503.20.

Less expense of ore bags, freight, hauling, etc., $4,450.79.

Net royalty, $10,052.41.

COST OF CONCENTRATION OF MILLING ORE.

During the experimental stage of milling, the drying of the ore, the conveying of it from one apparatus to the other, and the removal of the tailings from the mill had to be done by hand labor. The cost per ton of concentrate during such work was therefore somewhat higher than with the completed mill. All of the ore used during the experimental work was taken from the most convenient places on the Maggie C claim.

The cost of concentration in the completed mill averaged $20.41 per ton, delivered at the mill warehouse. The concentrate averaged about 3 per cent U_3O_8 content. The cost given does not include the cost of mining, tramming, hauling, and sorting the milling ore, and of delivery into the mill ore bin after resorting, which averaged $7.14 per ton of concentrates. The cost of mining and concentrating the mill ore, including all mill supplies, labor, repairs, amortization on mill equipment and construction, was $27.55 per ton of concentrate, delivered at the mill at Long Park. To this has to be added $8.63 royalty; the cost of ore sacks, freight, and hauling, amounting to $33.10; and also amortization of equipment, taxes, accident liability insurance, and cost of cooperation with the Bureau of Mines. The cost of concentrates produced during the first month's run was therefore about as follows: $57.28 per ton delivered at mill warehouse; $75.28 per ton delivered at railroad station, Placerville, Colo.; $84.28 delivered at the Denver, Colo., plant; or, based on its U_3O_8 content, $1.44 per pound of U_3O_8 in the concentrate, f. o. b. Denver plant.

[Figures followed by an asterisk indicate items that, by the agreement between the National Radium Institute and the Crucible Steel Co., were included in the "cost of treatment" in figuring the royalty.]

Calculation of cost of concentration of carnotite milling ore.

[Runs 1, 2, 3, and 4. See p. 74 for details of operation.]

				Cost of concentrates per ton.
Labor			$468.58	$12.890*
566 gallons of kerosene, at 25 cents per gallon			141.50	3.892*
57½ gallons of gasoline, at 35 cents per gallon			18.98	.522*
5½ tons of coal, at $8.50 per ton			43.52	1.197*
Oil, grease, water, etc			15.00	.412*
Repair parts and replacement of wearing parts on apparatus			54.53	1.500*
Chargeable directly to operation			742.11	20.413
Amortization on equipment and cost of cooperation			545.25	15.00
Accident liability insurance, and taxes			54.53	1.50*
			1,341.89	36.913
Cost of mining, sorting, picking, and hauling of milling ore to mill ore bin			259.38	7.135*
837 ore bags and twine, at 20 cents each			167.40	4.605
Cost f. o. b. mill, Long Park, Colo			1,768.67	48.653
Hauling to Placerville, Colo			654.30	18.00
Cost f. o. b. Placerville, Colo			2,422.97	66.653
Receiving and loading on cars at Placerville			18.18	.50
Freight to Denver, Colo			290.80	8.00
Unloading and sampling			18.18	.50
Royalty, 15 per cent on 2,125.1 pounds U_3O_8, at $2 per pound f. o. b. Denver		$637.53	2,750.13	75.653
Less 15 per cent of the following charges:				
Cost of treatment, at $29.05 per ton	$154.16			
Ore bags and twine	25.11			
Hauling to Placerville	98.14			
Receiving and loading at Placerville	2.73			
Freight	43.62			
		323.76		
Net royalty			313.77	8.631
Total cost of concentrate f. o. b. Denver plant			3,063.90	84.284

Cost per pound of U_3O_8 in concentrates $1.44 f. o. b. Denver plant, or $84.29 per ton of concentrates.

COST OF CONCENTRATION MILL.

The cost of the complete mill as built and operated by the National Radium Institute was $10,000, which covers the cost of all material and equipment, including freight charges and haulage, and the construction work. The figure given includes a small sum for additional expenditures made for experimental work and for apparatus which was discarded, as well as the expense incurred in reconstruction when the change to the completed mill was made.

PUBLICATIONS ON MINERAL TECHNOLOGY.

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DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

EXTRACTION AND RECOVERY OF RADIUM, URANIUM AND VANADIUM FROM CARNOTITE

BY

CHARLES L. PARSONS, R. B. MOORE, S. C. LIND

AND

O. C. SCHAEFER

NATIONAL RADIUM INSTITUTE
COOPERATIVE AGREEMENT

[This report has been prepared under a cooperative agreement with the National Radium Institute]

WASHINGTON
GOVERNMENT PRINTING OFFICE
1915

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First edition. November, 1915.

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EXTRACTION AND RECOVERY OF RADIUM, URANIUM, AND VANADIUM FROM CARNOTITE.

By Charles L. Parsons, R. B. Moore, S. C. Lind, and O. C. Schaefer.

CHAPTER I. INTRODUCTION.

Early in 1912, from information received by the Bureau of Mines, it became evident that quantities of valuable radium-bearing ore from Colorado were being exported for manufacture in foreign countries and that at least part of the manufactured product was being returned to this country and sold at a price entirely incommensurate with that received by the miner and the prospector. Consequently, in line with its efforts to increase efficiency in the mining and treatment of mineral resources, the Bureau of Mines undertook an investigation of conditions. Results of this preliminary investigation were published, in the summer of 1913, as Bulletin 70,[a] which gave an outline of the radium deposits of the West, the conditions of mining, the wastes involved, and the status of the production of radium from its ores.

At about this time newspapers and medical journals were publishing statements regarding the value of radium in cancer therapy. These reports, although many of them were exaggerated, brought out clearly the remarkable virtue of the gamma rays for the treatment of cancer and other malignant growths. In view of the fact that the largest known supplies of radium ore in the world are in the States of Colorado and Utah and on the public lands of the United States, it was deemed highly important for the Bureau of Mines to study the production of radium in detail. Such study, it was hoped, would enable the prospector and miner to obtain a fairer price for the ores and would furnish information necessary in case Congress appropriated funds to develop methods for extracting radium from ore on the public lands under Government supervision, to be supplied at cost to the hospitals of the Army, Navy, and Public Health Service.

[a] Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium Bull. 70, Bureau of Mines, 1913, 101 pp., 4 pls., 2 figs.

FORMATION OF NATIONAL RADIUM INSTITUTE.

At that time no appropriation for such work was available for this purpose. However, the bureau ascertained that Dr. Howard A. Kelly, of Baltimore, Md., and Dr. James Douglas, of New York City, were deeply interested in the production of radium for use in two hospitals with which they were closely connected. The suggestion was made them that they form a radium institute and endeavor to work up some of the American ores and keep the radium in this country for use among our own people. A member of the bureau went with Dr. Kelly to the Paradox Valley, Colo., and inspected the mines there, it having been agreed that if ore could be procured a radium institute would be founded and necessary funds furnished to work up the ore. As a result of this visit a conference was held with the officers of the Crucible Steel Co., who, through a subsidiary company, control 16 carnotite claims in Montrose County, Colo., which had been held pending the time when extraction of vanadium and uranium from the ore would be profitable. The company agreed that if the National Radium Institute should be formed these claims would be leased to the institute on a 15 per cent royalty basis under an agreement providing for the return of the uranium and vanadium content of the ore to the company. The National Radium Institute was duly incorporated and the claims were leased on the terms mentioned.

COOPERATIVE AGREEMENT BETWEEN THE NATIONAL RADIUM INSTITUTE AND THE BUREAU OF MINES.

Subsequently the institute proposed to the Bureau of Mines that a cooperative agreement looking to the development of the radium ores be made. The legality of the proposed agreement was carefully examined and the agreement was approved by the proper officials, many precedents for similar cooperative work having been found, especially in connection with arrangements between the Department of Agriculture and farmers throughout the country. In the light of the manifestly great usefulness of radium in therapy the importance of studying the ores and their extraction and of disseminating the resulting information was deemed so great and the acceptance of the institute's offer so highly desirable that, on October 13, 1913, the offer was accepted. The text of the agreement follows:

October 7, 1913.

The Director, Bureau of Mines.

Sir: The National Radium Institute (Inc.) is a corporation duly organized and existing under and by virtue of the laws of the State of Delaware, and is interested in the study of the best methods of producing uranium, vanadium, and radium, and in the mining and concentration of ores from which these products may be obtained for

use in radium therapy. It is interested in certain carnotite and other suitable ores mined in Colorado from which these products are obtainable.

For the benefit of the people, and to secure the advantage of a scientific and technologic study of efficient methods of production, as well as mining and concentration of the ores, without waste, the institute solicits the cooperation of your bureau, that is, the services, under your supervision, of enough chemists and mineral technologists connected with your bureau as may be necessary for the proper technical control and supervision of the mining and concentration of carnotite and other suitable Colorado ores, in the study of the methods outlined and generally in the prosecution of the investigation contemplated, the results to be published by you.

To aid in such work, the institute will furnish the sum of $150,000, to be available as follows and to be expended, after consultation with us, as you advise: $70,000 during the year ending August 30, 1914; $40,000 during the year ending August 30, 1915; $40,000 during the year ending August 30, 1916, it being estimated that about three years are required for the exhaustive investigation that is desired. The funds received from the sale of any by-products other than radium belong to us, and may also be used in the prosecution of this work. It is proposed to expend this money in the mining of the carnotite or other suitable ores, to an amount of 1,000 tons, more or less, of high-grade ore (over 2 per cent), or in the purchase of such ores as are necessary to an adequate study of the subject, and in the establishment and operation of a plant for the working up of said ores; also in the erection and operation of a small concentration plant for the treatment of any low-grade ore (less than 2 per cent U_3O_8) as may be mined or purchased—and of course in the payment of necessary wages of the plant operatives.

The institute is also entirely willing that any radium produced in excess of 7 grams of anhydrous radium bromide ($RaBr_2$) from any 1,000 tons of carnotite ore may be donated to the bureau for further experiment and study.

As the institute has already on hand 100 tons (more or less) of carnotite, it is made a condition of this offer that this ore be first extracted far enough to demonstrate to the satisfaction of the institute the practicability of the processes employed. Until this is done no more than $5,000 shall be expended on mining or concentration.

At the completion of this demonstration the undersigned may withdraw from this project and shall not thereafter be liable for any further expenditure of money other than sufficient to take up all obligations incurred up to the date of such withdrawal, and no further demands on account of said $150,000 shall be made. All expense incurred in such transaction shall, in any event, be on account of said $150,000.

The institute would esteem it a favor if its cooperation should be recognized on the cover and title page of any published report of the results of this investigation.

If this proposition is acceptable, will you not kindly advise the institute as soon as you conveniently can?

Respectfully,

NATIONAL RADIUM INSTITUTE (INC.),
By HOWARD A. KELLY, *President.*

WASHINGTON, D. C., *October 13, 1913.*

The SECRETARY OF THE INTERIOR.

SIR: The above letter is respectfully referred to you for your approval.

Very respectfully,

VAN. H. MANNING,
Acting Director.

Approved October 13, 1913:
LEWIS C. LAYLIN,
Assistant Secretary.

PERSONNEL OF RADIUM PLANT AND INVESTIGATIONS.

Under this agreement, Charles L. Parsons, Chief of the Division of Mineral Technology, was designated by the Director of the Bureau of Mines to take charge of the investigations. He assigned R. B. Moore, physical chemist, to the direct management of the plant subsequently erected at Denver, Colo.; K. L. Kithil, mineral technologist of the bureau, to the management of the mining and concention of the ores; and J. A. Davis, assistant mining engineer, to superintend the mining and concentration work at Long Park, Colo. O. C. Schaefer, who had had experience in chemical-engineering problems and plant construction, was made superintendent of the plant at Denver, and H. M. Eastman, junior chemist in radioactivity, was appointed assistant superintendent. To Mr. Moore was also assigned S. C. Lind, chemist in radioactivity, who had had long experience in the physical chemistry of radium, both at the laboratory of Mme. Curie, in Paris, and in the Radium Research Institute, Vienna. C. F. Whittemore, assistant chemist in radioactivity, and Messrs. J. C. Morgan, Alan Leighton, and H. A. Doerner, junior chemists in radioactivity, have rendered valuable assistance during the progress of the work, as did Prof. Herman Schlundt, who was employed during the summer of 1914 on radium measurements. The experimental work and the installation of apparatus for the electric-furnace production of uranium oxide were done by H. W. Gillett, alloy chemist, and J. M. Lohr, assistant alloy chemist. G. E. Bost, H. H. Barker, and J. E. Underwood, specialists in crystallizing radium, have been added more recently to the force. Each and all of the persons mentioned have loyally and efficiently served the bureau, and hearty acknowledgment of their work is made.

CONSTRUCTION OF PLANT.

Immediately after the signing of the agreement and the organization of the work, plans were begun for constructing a plant at Denver, and for mining the ore at Long Park, some 20 miles from Naturita, Colo. The plant was first built on an experimental basis, and after preliminary operation in the months of March, April, and May, 1914, began regular production of radium in June, 1914. The work was so successful that the directors of the National Radium Institute deemed it advisable to increase the capacity of the plant more than 100 per cent, and the construction of an addition was completed February 1, 1915. In addition, a grinding and sampling mill were built, as was a nitric acid plant, it having been found that a large part of the nitric acid could be recovered as sodium nitrate and used over and over again. The nitric acid plant has been in successful and continuous operation since February 1 of the present year and has greatly reduced the cost of producing radium salts.

The capacity of the plant is 3½ tons of ore per day, and the plant operations have been eminently successful. Up to October 10, 1915, nearly 5 grams of radium element had been extracted and approximately 2½ grams of element delivered in the form of radium bromide of such degree of purity as was desired by the hospitals concerned.

DEVELOPMENT OF EXTRACTION METHODS.

Entirely new methods, differing from former methods not only in detail but in apparatus used, have been devised for extracting radium from the ore, and for converting the sulphates to chlorides. The time required for the cycle of operation is probably much shorter than in any other plant and the recovery of radium is greater. Furthermore, it is believed that the costs of production are less than those of any other method used heretofore. It should be remembered that the methods apply solely to the treatment of carnotite ore. The bureau has been anxious to try to extract radium from pitchblende, the treatment of which would undoubtedly require some modification of the methods used for carnotite. Unfortunately, however, no pitchblende has been available, and it has been impossible to procure even so much as a ton of this material for experimental use. On the other hand, by the time this report has been issued nearly 1,000 tons of carnotite will have been treated.

Incidentally the methods for preparing sodium uranate, uranium oxide, and iron vanadate have been studied, and the necessary principles for procuring these in pure and marketable condition have been evolved. With ore obtained from Government land or produced at a cost as low as that maintained in the operations of the bureau, it has been shown that the cost of producing radium need not exceed $40,000 per gram and that the extraction of at least 90 per cent of the radium present may be obtained from good quality ore, such as the bureau has been able to procure. Full details of these methods, with a description of the plant, apparatus, and chemical-engineering problems involved, appear in the following pages.

A separate report on the mining and concentration of radium ores is being prepared and will shortly be published by the bureau.

In this general summary there are three points on which it is felt that the position of the Bureau of Mines should be made clear—first, the application of radium to the cure of cancer; second, the market or selling price of radium; and third, the object of the bureau in undertaking this investigation.

USE OF RADIUM IN CANCER THERAPY.

The application of radium to the treatment of cancer and other malignant growths has made much more rapid progress in the past two years than is generally appreciated even by the medical profes-

sion. All kinds of opinions may be easily had from the laity and from cancer experts, and the public can draw almost any conclusion from the statements appearing in newspapers, but preference should be given the opinions of those persons who have had extended experience in the use of radium and in sufficient quantities to entitle them to speak authoritatively. One thing regarding radium therapy that has been developed during the past two years is that if the gamma rays of radium are to be successfully applied to the cure of cancer a much larger quantity of radium must be used than is possessed by not more than two or three individual surgeons or hospitals in the United States. The bureau knows of no individual or hospital that has had the privilege of working with so much as half a gram of radium that does not report cures in increasing number or that is not able to treat even advanced cases with increasing assurance of success as a knowledge of technique is developed. Also, the extent and variety of the cures reported and the helpful effect of the gamma rays at depth are almost in direct proportion to the quantity of radium that can be applied at one time to the patient. Blacksmiths working with hand forges would have as much right to condemn the smelting of iron in 500-ton blast furnaces as physicians or surgeons working with a few milligrams of radium to condemn its application in quantities of 500 milligrams or more. This statement does not mean that radium in comparatively small amounts may not be effective with some forms of cancer or some malignant growths in their incipiency, but in two American hospitals that have 500 or more milligrams of radium results are being obtained daily that unquestionably will not be duplicated elsewhere until equal quantities are available.

MARKET PRICE OF RADIUM.

The market price of radium for some two or three years has been $120,000 and up per gram of element according to purity. At the time of the beginning of the European war there was 16 to 20 grams of radium contracted for abroad, chiefly in Germany, at prices above $120,000 per gram. Seemingly the war has not affected the market price but simply the amount produced. As to the selling price of radium in the future, the bureau makes no prediction, but it does not follow that the selling price will decline because of the development of cheaper methods of production. Rather is the case somewhat analogous to the production of gold, diamonds, or any other material that is in steady demand but occur only in very small quantities. Beyond doubt, the amount of radium in nature is exceedingly small. Other deposits of radium ores may be found, but it is highly improbable that the rarity of uranium ores will ever be greatly modified, so that the price of the finished material will largely depend on the ability to procure the raw material.

According to the best evidence that the bureau can obtain, and it has investigated the matter carefully, there is not sufficient ore available to maintain for many more years even the rate of production of 1914. In this connection it should be remembered that the radium produced by the National Radium Institute is not for sale nor for distribution. The Radium Institute was organized for the purpose of studying the curative properties of radium and not for private gain. The radium produced is being used in two hospitals—the Howard A. Kelly Hospital in Baltimore and the General Memorial Hospital in New York.

OBJECT OF INVESTIGATIONS.

In undertaking the radium investigations the bureau had two objects in view—first, to enable the miner and prospector to obtain a just return for the ores and, second, to convince the people that radium ores on the public lands of the United States should be mined and treated under Government supervision and the radium placed in the hospitals of the Army, Navy, and Public Health Service for the benefit of the people of this country.

At the time the bureau began its radium investigations, the miners and prospectors were receiving approximately only $75 to $80 per ton for ore containing 2 per cent uranium oxide, this price being for ore delivered f. o. b. New York in transit to Europe. The ore was being sent abroad in quantity and was not being treated in this country, although one or two companies were rapidly purchasing from prospectors available claims at $50 to $200 per claim, and thereby laying the foundation of a future possible monopoly in this rare material. It was hoped that methods could be found that would enable miners or associations of miners to extract the radium from their own ores, place it on the market, and obtain profits that had been going to foreign manufacturers. By August, 1914, when the European war came on, ore was selling for a much higher price than had been obtained in the summer of 1912.

The methods of extracting the radium that have been evolved and are described herein may be utilized under intelligent technical control by any firm, association, or corporation that can obtain the raw material either by purchase or from its own claims. These methods are being patented for the benefit of the people, and may be freely used by anyone within the United States.

It was also the desire of the bureau to show beyond doubt that, if authorized by Congress, the radium could be procured from ores now owned by the people at a cost approximating one-third the market price, and that it was much better policy for the Government to obtain the radium so greatly needed in the hospitals of the Army, Navy, and Public Health Service at a cost not exceeding one-third the present

market price than to give these ores to large corporations and then purchase as radium salts material that had belonged to the people, at a price insuring large profits to the possessors.

Unfortunately the bill introduced into Congress in January, 1914, that would have accomplished this purpose failed to pass, and in the meantime the carnotite areas have been thoroughly prospected and the most valuable claims located. Accordingly the bureau must be content with the good that may come to the American people through the radium being extracted in the plant of the National Radium Institute until such time as the Government may furnish funds sufficient to enable it to purchase in the open market such ore as may be necessary to supplement the little that remains unlocated on Government land.

CHAPTER II. DIFFERENT METHODS OF TREATING RADIUM ORES, INCLUDING NITRIC ACID METHOD.

Until a year or two ago little had been published on methods of treating any of the uranium ores for the extraction of radium, although a number of patents have been issued both in this country and abroad. Therefore persons interested in methods of extracting radium have had little to guide them, as details of the methods in use have been closely guarded. The effect has been rather to discourage those who might have been inclined to invest money in the production and sale of radium. Consequently, there is a real need of the presentation of all of the necessary details in the successful treatment of uranium ores, and the refining of the finished products.

The different methods heretofore used may be classified under three general heads, as follows: (1) Use of an acid leach; (2) use of an alkaline leach followed by an acid leach; (3) fusing the ore with some material that will break up the ore and make the extraction of the valuable contents possible. These different methods have been discussed at some length in Bulletin 70,[a] but are described here in more detail.

USE OF ACID LEACH.

In the method involving the use of an acid leach, the two acids employed up to the present are sulphuric and hydrochloric.

LEACHING WITH SULPHURIC ACID.

A number of patents have been issued in connection with the use of sulphuric acid. The Fleck, Haldane, and White patent[b] claims that radium, uranium, and vanadium can be extracted successfully in the following way:

METHOD OF FLECK, HALDANE, AND WHITE.

The ore is crushed, preferably to 20 to 40 mesh, by any suitable means, and is agitated with hot sulphuric acid of 15 to 20 per cent concentration, the proportion of acid used depending upon the quality of the ore. As a rule, 400 pounds of 65° B. sulphuric acid diluted to 15 to 20 per cent is sufficient for the treatment of 1 ton of ore. The resulting acid solution contains the uranium, vanadium,

[a] Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium: Bull. 70, Bureau of Mines, 1914, pp. 69-82.

[b] United States patent 890584.

and copper, if copper is present, and is preferably filtered or otherwise clarified. The resulting clear, acid solution is then brought into contact with fresh ore, heated, and agitated, the solution being thereby neutralized. At the same time, a part of the uranium, vanadium, and iron is precipitated on the ore as basic sulphates or carbonates, the effect of this precipitation being to enrich the ore, which may be of low grade. If necessary, the neutral solution is again clarified, and constitutes a part of the stock solution suitable for further treatment. The enriched ore that has served for the neutralization of the acid solution, either alone or mixed with fresh ore, is treated with sulphuric acid, as described above, yielding an acid solution, which, after neutralization, is added to the stock solution.

The ore residues from the treatment with sulphuric acid, as well as the residues from the similar treatment of the enriched ore, are freed from remaining values by being washed with dilute sulphuric acid. The resulting acid washings are then strengthened by the addition of sulphuric acid until they contain, preferably 15 to 20 per cent of the acid, and are utilized for continuing the process.

The nearly neutral stock solution, containing uranium, vanadium, iron, and possibly copper, is then treated with sulphurous acid by subjecting the solution to the action of sulphur dioxide obtained by roasting sulphur or sulphide ores. This treatment reduces the iron and the vanadium. Reduction of the iron to the ferrous state is an advantage, because in the subsequent precipitation of the uranium and vanadium less iron is precipitated, and the valuable metals are therefore in a more concentrated form. The sulphuric acid from the sulphur dioxide is also available for the treatment of additional quantities of ore.

The reduced solution is separated from the ore by filtration or decantation, and is treated with such a quantity of finely pulverized limestone as calculation has shown will bring it to the point at which uranium, vanadium, and copper will just begin to be precipitated, calcium sulphate being formed. The solution is then separated from the calcium sulphate, and sufficient pulverized limestone is added to cause complete precipitation of the uranium and vanadium when the liquid is boiled. The precipitate, which comprises a complex mixture containing basic sulphates and carbonates of uranium and vanadium, compounds of iron, and hydrated calcium sulphate, is at first green, but in air changes rapidly to light green or yellow. This precipitate may be used as the final uranium-bearing product, or it may be further refined or concentrated by any known or suitable method, as the radium, of course, remains with the insoluble residue.

The residue may be agitated with water, and after the larger part of the coarse material has settled, the slimes may be separated and

allowed to settle. After the decantation of the clear liquid, the residue, consisting largely of calcium sulphate, barium sulphate, and finely divided silica, constitutes the radium concentrate. From this concentrate the radium may be extracted and refined by any suitable method. By the procedure outlined, it is possible to get a radium concentrate carrying 50 to 100 milligrams of radium per ton of concentrate, the exact proportion depending on the grade of the ore treated.

OTHER METHODS.

The other methods of leaching with sulphuric acid are more or less similar to that of Fleck, Haldane, and White. Some involve a preliminary roasting of the ore before treatment with acid.

McCoy's method [a] involves mixing the ore with either dilute or concentrated sulphuric acid to form a mud, and roasting, at first to a temperature of 100° C. and finally to 300° C. or even higher. When the roasted material is leached with water, the iron, vanadium, and uranium are dissolved, and the radium remains with the insoluble residue from which it can be recovered by any suitable means.

In all such methods of leaching the radium remains with the insoluble residue, and usually some sliming method is used to obtain it in a more concentrated form.

If carnotite is treated with hot concentrated sulphuric acid, the radium, instead of being precipitated, is dissolved. If the solution obtained from this treatment is diluted and allowed to stand, radium and barium are precipitated, more barium chloride being added, if necessary. In order to get a good extraction it is necessary to use two to three times as much acid as ore.

Theoretically, it might be possible to get radium into solution by passing carbon dioxide into water in which carnotite ore is suspended. This method, however, does not give satisfactory results.

LEACHING WITH HYDROCHLORIC ACID.

Hydrochloric acid has been used for leaching in a number of commercial plants. The ore is boiled with hydrochloric acid—at some plants with the strong acid, and at others with weak acid—one part of strong acid to two parts of water. As compared with sulphuric acid, hydrochloric has the advantage of putting the radium into solution at the start. Also, as hydrochloric acid has a reducing action on the vanadium, it acts as a good solvent for that metal as well as for the uranium. If strong acid is used, the radium can be precipitated from the acid solution by partial neutralization either with sodium hydroxide or sodium carbonate, preferably the former,

[a] United States patent 1098282.

and the addition of sodium sulphate and barium chloride if the ore does not contain sufficient barium. After[b] the separation of the precipitated radium-barium sulphate, the uranium and vanadium can be recovered by any suitable process.

USE OF ALKALINE LEACH.

It is probable that some of the early experimental work on extracting radium from carnotite ore involved the boiling of the ore with sodium carbonate, thereby getting rid of most of the uranium and part of the vanadium. The residue, after having been thoroughly washed, was then leached with dilute chemically pure hydrochloric acid in order to extract the carbonates present, of which radium would be one. The Haynes-Engle process [a] for the recovery of uranium and vanadium covered this method, but the patent specifications do not show what is necessary for the final recovery of the radium, as at the time the patent was issued carnotite was mainly valuable for uranium and vanadium and not for radium.

HAYNES-ENGLE PROCESS.

According to the Haynes-Engle patent the ore is first crushed to 12 mesh and is then boiled with a solution of an alkaline carbonate, preferably sodium carbonate or potassium carbonate, until the uranium or vanadium, or both, in the ore is dissolved. The strength of the sodium carbonate solution and the length of time necessary to boil it are determined by the proportion of uranium and vanadium in the ore, and will probably vary considerably. The originators of the process claim, however, that 100 pounds of sodium carbonate per ton of ore for each 1 per cent of uranium and vanadium, or either, present will give good results. The time required for boiling should be about one hour. After the uranium and vanadium, or either, has been dissolved, the clear solution is drained into a separate tank. The uranium is precipitated as sodium uranate by the addition of sodium hydroxide to the solution. This precipitate is removed from the solution, which contains all of the vanadium. From the solution, either with or without neutralization, the vanadium is precipitated as calcium vanadate by the addition of water-slaked lime.

It is claimed that an extraction of 80 per cent of the uranium and 60 to 65 per cent of the vanadium can be obtained by this process.

BLEECKER'S PROCESS.

What was lacking in the Haynes-Engle process has been covered by a patent issued to Warren F. Bleecker.[b] After the ore has been boiled with sodium carbonate, as in the Haynes-Engle process, and

[a] United States patent 808839. [b] United States patent 1065581.

washed, the residue is leached with a dilute acid, preferably hydrochloric or nitric. This treatment of the ore dissolves the radium, forming radium-barium chloride, and the solution can be drawn off and stored for any approved treatment to recover the radium. If desired, the ore may be leached once more with acid, 10 per cent sulphuric acid being used, the object being to recover the vanadium not extracted by the first leaching.

By Bleecker's method not only is nearly all of the uranium and part of the vanadium recovered but the radium is also obtained in soluble form. As the bulk of the liquid would be too great for direct fractionation, it would be necessary to precipitate the radium and barium salts in solution by adding a sufficient quantity of sulphuric acid or sodium sulphate. By this means, practically all of the radium in solution could be recovered as radium-barium sulphate, which would need further treatment, as described in a subsequent chapter.

Another patent [a] issued to Bleecker covered some modifications of the process outlined. Under the later patent the ore is boiled with a solution of an alkaline hydroxide, and, if desired, an alkaline carbonate may be added to the solution. For example, sodium hydroxide and sodium carbonate may be used, in which event the resulting liquid will be a solution of sodium vanadate, from which the vanadium may be recovered by any approved process. After the sodium hydroxide in the ore has been neutralized with acid, the ore is leached with a solution of alkaline carbonate, the temperature of the solution being kept at about 90° C. This treatment will dissolve the uranium as the soluble sodium uranyl carbonate. The solution can be subsequently treated by any suitable process for the recovery of the uranium. After the ore has been thoroughly washed to free it from soluble carbonates and sulphates, it is leached with an acid other than sulphuric, then washed with water. Eight per cent hydrochloric or nitric acid gives good results. By this method the radium is extracted as the soluble chloride in conjunction with barium, calcium, etc., and can be precipitated from the solution, as already described, by the addition of sulphuric acid or sodium sulphate.

EXTRACTION OF RADIUM BY FUSION METHODS.

FUSION WITH SODIUM SULPHATE.

The first method that was used in connection with the extraction of radium was that of fusing pitchblende ores with sodium sulphate. This method was originally used by the Austrian Government in connection with the treatment of the pitchblende ores from the Government mines at St. Joachimsthal. By this fusion the uranium in the

[a] United States patent 1068730.

ore is changed to sodium uranate, which can be dissolved from the insoluble residue, after leaching with water, by means of dilute sulphuric acid. The radium remains behind with the residue, and was, before the discovery of radium, discarded.

PROCEDURE OF HAITINGER AND ULRICH.

The extraction of the radium from this residue is well described by Haitinger and Ulrich,[a] and, with some minor changes, is probably similar to the method now used by the Austrian Government. The work was done in the laboratory of the Austrian Incandescent Gas-light & Electric Co. Ten thousand kilograms (22,000 pounds) of pitchblende residues was treated, representing about 30,000 kilograms (66,000 pounds) of pitchblende containing 53.4 per cent of $U_3\ O_8$. Moisture in the material in the various shipments varied from 10.3 to 18.4 per cent. The moisture was driven off at a temperature of 105° C., the 10,000 kilograms of residue losing 1,340 kilograms in this way. The work took two years, owing to time spent on analyses at the start and to experiments to obtain the best results. The method given was developed as being most suitable to the equipment available. Five thousand kilograms annually could be treated.

The chemical operations involved were, first, the decomposition of the residues; second, the removal of the radium sulphate by precipitation; and third, the solution and concentration of the radium sulphate. The first step consisted of digesting the residue with sodium hydroxide, 100 kilograms being boiled for one day with a solution of 50 kilograms of hydroxide in 200 liters of water. Forty per cent of the alkali was converted to sulphate and to silicate. The solution contained some radium, but the total radium so dissolved from the entire 10,000 kilograms of residue represented the radium equivalent of only 10 kilograms. Therefore this solution was thrown away.

After the boiling the residue was allowed to settle and the liquid was decanted. The residue was washed to remove the greater part of the sulphates, filtering and washing being done in a funnel with a capacity of 100 kilograms and having a lead suction tube 3 meters long. The vessel containing the washed residue was placed on a water bath, and the residue was treated with an equal weight of dilute (1:1) crude hydrochloric acid. After prolonged heating the acid solution was decanted and the residue washed with water. The water was then used to dilute the next part of acid for a new sample.

Crystals of calcium sulphate and lead chloride formed in the acid solution as it cooled. Neither the solution nor the crystals contained

[a] Haitinger, Ludwig, and Ulrich, Karl, Bericht über die Bearbeitung der Pechblend-Rückstände: K. K. Akad. Wissenschaft., vol. 117, 1908, p. 619.

an appreciable quantity of radium, but nearly all of the polonium and of the actinium was in the solution. The solution was therefore treated with ammonium hydroxide to precipitate the polonium and actinium. The filtrate was not radioactive and was discarded.

The residue from the treatment with crude hydrochloric acid was boiled in a solution of sodium carbonate, the carbonate, made by the ammonia process, being free from sulphates. Fifty kilograms of sodium carbonate in 200 liters of water was used for a 100-kilogram sample. By this treatment a large part of the radium sulphate was converted to radium carbonate. Therefore, in subsequent treatments the solutions had to be kept free from sulphate. The residue was washed till free from all trace of sulphate and then treated with pure hydrochloric acid. The boiling with soda and the treatment with acid was repeated three times. After the third treatment only 2 per cent of the original radium content remained in the residue, which was thrown away. The soda extracts were practically free from radium. The washing of each of the various residues consumed four to six weeks. The hydrochloric acid extracts, containing nearly all the radium, were united and the radium was precipitated as a sulphate with sulphuric acid. Besides the radium, the precipitate contained, of course, the alkali earths, including calcium, much lead containing radio-lead, and a small quantity of rare earths containing actinium. The sulphate precipitate, called crude sulphate, represented 0.5 to 2 per cent of the weight of the original residue.

The crude sulphate was reconverted to carbonate by repeated boiling with sodium carbonate solution. All of the sulphate from any one sample could not be converted; therefore, the final residue from the treatments was not thrown away, but added to a new part of crude sulphate. After each carbonate treatment, an extraction with hydrochloric acid was made. The lead chloride formed in the solutions was removed and freed from radium by repeated crystallization in hot water. Sixty kilograms of lead chloride was thus obtained from the entire 10,000 kilograms of residue. It was saved and treated for its content of radio-lead.

The hydrochloric acid solutions from the crude sulphate were freed completely from lead by hydrogen sulphide and were then evaporated to dryness on a steam bath. The calcium chloride in the residue so obtained was dissolved in concentrated hydrochloric acid, in which barium chloride is only slightly soluble and radium chloride is still less soluble. The residue remaining, called crude chloride, consisted of radium and barium chlorides, with some strontium and calcium chlorides and traces of other impurities.

From this point on the concentration was continued by fractional crystallizations from water solutions. Radium chloride, which is the least soluble of the chlorides, accumulated in the crystals, the foreign

matter remaining more and more in the mother liquor. The first fraction was, of course, the richest in radium. Two steps in this process were of particular importance—first, the separation of as large a quantity of radium-free barium chloride as was possible; second, the making of a relatively large first fraction. The second step could be satisfactorily attained by temporarily stopping the crystallization of the first series until the crystals of the second series were of sufficient radioactivity to be united with the first.

The crystallizations were all carried out on a steam bath in order to avoid contamination with sulphate, as might have occurred if heating had been done with a direct flame.

Finally, two portions of crystals were obtained, one of about 2 kilograms, containing nearly all of the radium, and the other of about 11 kilograms, containing little radium.

The 2-kilogram portion was treated as raw material for the production of radium chloride free from barium. After this portion had been crystallized about 30 times, the first fraction of about 9 grams was further crystallized, and the lower fractions were combined into three groups according to their activity. The 9-gram portion was first purified with hydrogen sulphide, which removed traces of lead that probably came from the glassware. Further work was conducted in quartz vessels. The salt was dissolved in dilute hydrochloric acid, warmed, and allowed to crystallize. Four fractions were so obtained.

Atomic-weight determinations were made with three of the fractions, the values obtained being 143.2, 185.2, and 225. The latter represented practically pure radium chloride.

Some of the lower fractions of barium chloride that were poor in radium were converted to the bromide and then fractionated. Only one portion, that which should contain the most radium—that is, the last of the four analagous fractions—was saved. The other portions were reconverted to chloride and added to the main chloride crystallization system. In all, 3 grams of pure dry radium chloride and 0.236 gram of radium bromide were obtained from 10,000 kilograms of original residue.

RADCLIFFE'S METHOD.

Although Haitinger and Ulrich's method is suitable for the treatment of pitchblende, which contains much more uranium oxide than does the average grade of carnotite, as shipped, the method would not be applicable to the latter ore. The same statement applies to the method that has been used by Radcliffe,[a] as the ore that his method is designed to treat is widely different from the carnotite in Colorado and Utah. The actual mineral carrying the uranium that is found in

[a] United States patent 1049145.

Olary, South Australia, is carnotite, but it is associated with considerable quantities of ilmenite and rare-earth minerals that are not found in American carnotite.

The dry ore is crushed to pass a 20-mesh sieve, and is then concentrated magnetically; the concentrates amount to about 30 per cent of the original ore, and, as they are insoluble in acids, a fusion process is necessary to effect the initial decomposition. In the fusion process the concentrates are mixed with three times their weight of salt cake and fused in a reverberatory furnace. The fused product is crushed to pass an 8-mesh sieve and is put into wooden vats fitted with agitators. Cold water is fed continuously into the vats at the bottom, and an overflow is provided near the top. By suitable adjustments it is possible to separate out on the bottoms of the vats a considerable amount of comparatively coarse material that is almost free from radium and uranium. The turbid liquid overflowing carries in suspension the radium, lead, and barium as sulphates, together with a considerable amount of finely divided silica. In solution are found the uranium, the rare earths, and part of the iron and "acid earths" contained in the ore. The coarse residues are removed daily, rewashed, and discarded.

The overflow from the dissolving vats is pumped to large lead-lined tanks and allowed to stand all night. The "slimes" settle completely in 12 hours and the clear liquid is drawn off daily and treated for the recovery of uranium. The slimes, which constitute when dry approximately 10 per cent of the weight of the concentrates, are collected weekly and treated for the recovery of radium.

The clear solution, containing the uranium and much of the iron and other bases, together with a large amount of sodium salts, is fed into a series of vats containing a measured excess of a mixture of carbonate and bicarbonate of soda. This is heated and agitated by means of steam jets; the iron and most of the other bases are precipitated, and the uranium, together with some of the rare earths, goes into solution. The iron is filtered off and the uranium solution is made barely acid with sulphuric acid and is heated, the carbon dioxide being expelled by a current of air. The uranium is then precipitated by the addition of ammonia. The ammonium uranate thus formed is thickened in conical settling tanks and in a hydroextractor. The pulp is dried and dehydrated in large muffles. The dried product is broken up and washed repeatedly with hot water to remove sodium salts, the final product containing about 75 per cent U_3O_8.

To recover the radium from the insoluble residue or slimes from the settling tank, the residue is mixed with half its dry weight of strong sulphuric acid and the liquid is allowed to stand for several days.

The residue is then washed by decantation and by the use of a vacuum filter. This treatment reduces the bulk considerably, dissolving out large amounts of iron salts and "acid earths." The washed slime, in quantities of about 200 kilograms, is then boiled in large steel boilers with an excess of a 20 per cent solution of sodium carbonate, the solution being replaced once during the boiling. This treatment dissolves a large amount of silica and converts much of the lead, radium, and barium sulphates to carbonates. The slime is then washed until the wash water gives no reaction for sulphates, and is then fed into a warm dilute solution of hydrochloric acid, agitated for a couple of hours, and allowed to settle all night. The clear solution is siphoned off, and lead, barium, and radium precipitated as sulphates. After having been washed once by decantation, the slime is again treated as above described, two treatments being sufficient to extract most of the radium.

The crude sulphate obtained in this manner is fused with carbonate of soda in large graphite pots and the product digested with hot water. The insoluble residue, after most of the metallic lead has been removed, is thoroughly washed and heated with hydrochloric acid, the solution is evaporated to dryness to dehydrate the silica, and the residue is moistened with acid and digested with hot water, the silica being filtered off.

FUSION WITH SODIUM CARBONATE.

Another method of radium extraction, that has been used by at least one company, is that of fusing the ore or carnotite concentrates with sodium carbonate; about three times as much carbonate as ore being used. The mixture is strongly heated in a reverberatory furnace lined with magnesite brick, and the fused mass is run directly into vats, in which it is leached. In this manner the silica is converted into sodium silicate and goes into solution together with the uranium and vanadium. The iron, calcium, barium, radium, etc., remain as the insoluble residue, which is washed in filter presses. This material is then treated with dilute sulphate-free hydrochloric acid, which dissolves the carbonates, and the radium and barium are precipitated by the addition of the requisite amount of sulphuric acid or sodium sulphate. The whole is allowed to settle in settling tanks, and the clear liquid drawn off, the barium-radium sulphates, mixed with a considerable amount of silica and other impurities, being left as a sludge at the bottom of the tank. This is taken off without previous filtration and dried, forming a crude radium-barium sulphate, which is then refined by a special process involving fractionation from neutral solution.

GENERAL DISCUSSION OF PROCESSES DESCRIBED.

As regards these different processes of radium extraction, it can be plainly seen that each has some disadvantages. Any process that involves the use of sulphuric acid as a leaching agent at once puts the radium in an insoluble form. Although a concentration is usually obtained which may run as high as 10 to 1, or even higher, the advantage of such a concentration is more than overcome by the disadvantage of having to treat the radium as an insoluble rather than a soluble product. The sliming method gives a concentrate consisting largely of calcium sulphate mixed with fine silica, the whole carrying a certain proportion of barium and radium sulphates. Owing to the presence of the silica, none of the short methods for recovering the radium from the insoluble sulphate can be used, and it is necessary to boil the material with sodium carbonate and then leach with chemically pure hydrochloric acid.

The labor required is considerable and makes the cost of refining high. The fact that this method of concentration could be used at the mines was used to justify the extra expense, but as it is cheaper to transport ore than sulphuric acid, the argument does not hold. In addition, the uranium and vanadium concentrate is in an undesirable form which must be retreated before final use. Any process that converts the radium in carnotite into an acid insoluble product, when it is already in the ore in a more easily treated condition, is open to serious criticism.

LEACHING WITH HYDROCHLORIC ACID.

Leaching with hydrochloric acid will prove successful with some carnotite ore, provided the acid is pracitically free from sulphuric acid, or contains less than 0.05 per cent of the latter acid, and provided the ore itself is exceedingly low in gypsum and other sulphates. In other words on a selected ore a hydrochloric-acid leach will prove reasonably satisfactory, and an 80 or even a 90 per cent extraction can be obtained under favorable conditions. The preferable concentration is an acid containing about 20 per cent hydrogen chloride, and a weight equal to that of the ore should be used. The filtration must take place while the acid is hot, and, therefore, must be rapid. The residue should be washed with more dilute acid and then with distilled water. Although a satisfactory extraction is frequently obtained, on the other hand the extraction from many ores may be as low as 50 or even 40 per cent, and the method can not, therefore, be considered satisfactory for universal use with carnotite ores.

FUSION WITH SODIUM CARBONATE AND LEACHING WITH ACID.

The treatment with sodium carbonate, followed by an acid leach, does not have the same objections. Indeed, this method can be used efficiently for the extraction of radium, the objections being more of a mechanical than of a chemical nature. When the ore is boiled with sodium carbonate, about 80 per cent of the uranium and possibly 60 per cent of the vanadium goes into solution. A previous roast or the use of oxidizing materials during the leaching is advantageous, if not necessary. The sodium carbonate solution can be separated either by the use of a filter press or by settling and decantation, as the liquor does not readily filter by gravity through ordinary filtering media. The ore thus treated must be thoroughly washed with distilled water, in order to remove as much as possible of the sodium silicate and sodium sulphate formed in the reaction. After this washing it is necessary to use chemically pure hydrochloric acid, or at least an acid absolutely free from sulphates, for the presence of small amounts of iron or other impurities is not injurious.

It is almost impossible to filter the acid solution after treatment with hydrochloric acid, because small quantities of sodium silicate remain with the ore, and the liberated silicic acid clogs any filtering medium that may be used. Experiments by the National Radium Institute showed that great difficulty would be experienced in filtering at this stage either by gravity, by suction, or by pressure. Again, settling and decantation might prove successful, but would involve the use of a large bulk of dilute acid. During the time required for settling, the radium would tend to precipitate, especially in the presence of silica. The process also involves the handling of the ore twice, but the main difficulties are in connection with filtration or the separation of the liquors from the residues.

After the acid solution has been removed, the radium can be reprecipitated as radium barium sulphate by adding to the acid solution the required amount of barium chloride and either sulphuric acid or sodium sulphate. The radium barium sulphate can then be obtained by settling and decanting the clear liquid from the precipitate, and finally removing the precipitate through an earthenware filter.

Whether the patent covering this process contains any new discovery might be questioned. Moreover, the same results can be obtained by using sodium bicarbonate instead of sodium carbonate.

FUSION WITH SODIUM CARBONATE.

The fusion methods used for pitchblende and the Austrian ores do not apply to American carnotite, although they may have some advantages for the particular ore they are designed to treat. The fusion of carnotite with sodium carbonate has some advantages and

some serious disadvantages. The main advantage is that it is adapted to carnotite in any form whether it be ore of 20 or 30 mesh or concentrates of more than 200 mesh. In addition the presence of sulphates in the ore is not deleterious, as they are removed at the same time as the sodium silicate. The disadvantages, however, are numerous. In the first place, as a large bulk of sodium carbonate has to be added to the ore considerable material has to be handled. After the fusion 3.5 tons of fused material are handled for 1 ton of ore or concentrates. The uranium and vanadium is in the filtrate from the leached material, and the presence of so much sodium carbonate and sodium silicate makes the cost of the recovery of the uranium and vanadium excessive.

The uranium can be recovered in one or two ways—either by adding sodium hydroxide directly to the filtrate, or by making the filtrate slightly acid with sulphuric acid and then adding sodium hydroxide to the hot liquid. In the first case the amount of sodium hydroxide required to precipitate the uranium in the presence of so much sodium carbonate is large and usually involves the precipitation also of a considerable amount of silica. The result can be more easily accomplished by first making the solution acid, but this involves again the use of a large amount of sulphuric acid. In addition the crude barium sulphates obtained are mixed with a considerable amount of silica, which makes them difficult to treat by ordinary methods.

It is doubtful whether a total recovery of radium of more than 70 per cent has been obtained by this process. This low recovery and the necessarily high costs more than counterbalances the advantages that the method may have.

POSSIBLE NEW METHODS OF RADIUM EXTRACTION.

In the search for a better method than those outlined the main object to be borne in mind is a high extraction and recovery of the radium. The importance of a high recovery of uranium and vanadium has been, to some degree, unduly emphasized; the main object is to obtain the radium.

ADVANTAGES OF USING NITRIC ACID.

For extracting the radium the use of nitric acid possesses many advantages, especially if the initial cost of the nitric acid can be reduced through the recovery of sodium nitrate as a by-product. The cost of the treatment would be very largely increased if 6 to 7 cents per pound of 100 per cent nitric acid had to be paid instead of the lower cost of hydrochloric acid. On the other hand, if, in connection with the nitric acid method, sodium nitrate can be recovered with small loss and nitric acid be once more made from the sodium

nitrate a cycle would be obtained that would reduce the cost of the nitric acid to a figure as low as, if not lower than, that for the hydrochloric acid.

In addition, the solvent action of nitric acid on radium sulphate is much greater than that of hydrochloric acid. Radium belongs to the calcium, strontium, and barium group, and, as analysts know, in order to precipitate barium sulphate completely and efficiently free nitric acid must be removed. Consequently, nitric acid has a much greater solvent action on radium even though the latter may be associated in the ore with reasonable quantities of sulphates.

This greater solvent action of nitric acid was the general basis for the method devised by the Bureau of Mines and used in the plant of the National Radium Institute. It has recently been recognized by Plum,[a] who suggests that possibly the best method for treating carnotite is to boil the ore first with sodium carbonate, leach the washed residue with hydrochloric acid, and follow this leaching with nitric acid in order to dissolve the 10 per cent of radium that he was unable to extract with hydrochloric acid. Plum had in mind the extraction of both radium and other radioactive constituents, such as polonium and actinium, but he plainly indicates that nitric acid can dissolve out of carnotite radium that can not be dissolved by hydrochloric acid, even after the ore has received a preliminary leaching with sodium carbonate. This finding was also established in the preliminary cooperative work carried on two years ago which led to the adoption of the nitric acid method.

When the use of nitric acid at the start will give just as high an extraction as the combined use of sodium carbonate, hydrochloric acid, and nitric acid, there is no need to use the three steps unless it is desired to recover the other radioactive constituents in the ore.

By the method he outlined, Plum was able to recover 89.9 per cent of the radium in the ore on a laboratory scale, using 1 kilogram of material; this extraction has been exceeded a number of times on carload lots in the plant of the National Radium Institute by the use of nitric acid alone. As a scientific accomplishment the recovery of the polonium, actinium, and ionium would be strongly advisable, but as these now have little or no commercial value they can not be taken into consideration in a commercial process.

In the nitric-acid process the radium is at once recovered as a high-grade radium barium sulphate, practically free from silica, and easily treated by improved methods. The process is adapted to recovering either the radium by itself, or the radium, uranium, and vanadium, as the radium is obtained first, and from that point all

[a] Plum, H. M., The extraction and separation of the radioactive constituents of carnotite: Jour. Am. Chem. Soc., vol. 37, 1915, p. 1797.

the other products may be discarded without further treatment, if so desired.

The different steps are each completed in one day, the equipment is not expensive, the extraction and recovery are high, and it is believed that the costs are lower than those with any other process for treating carnotite.

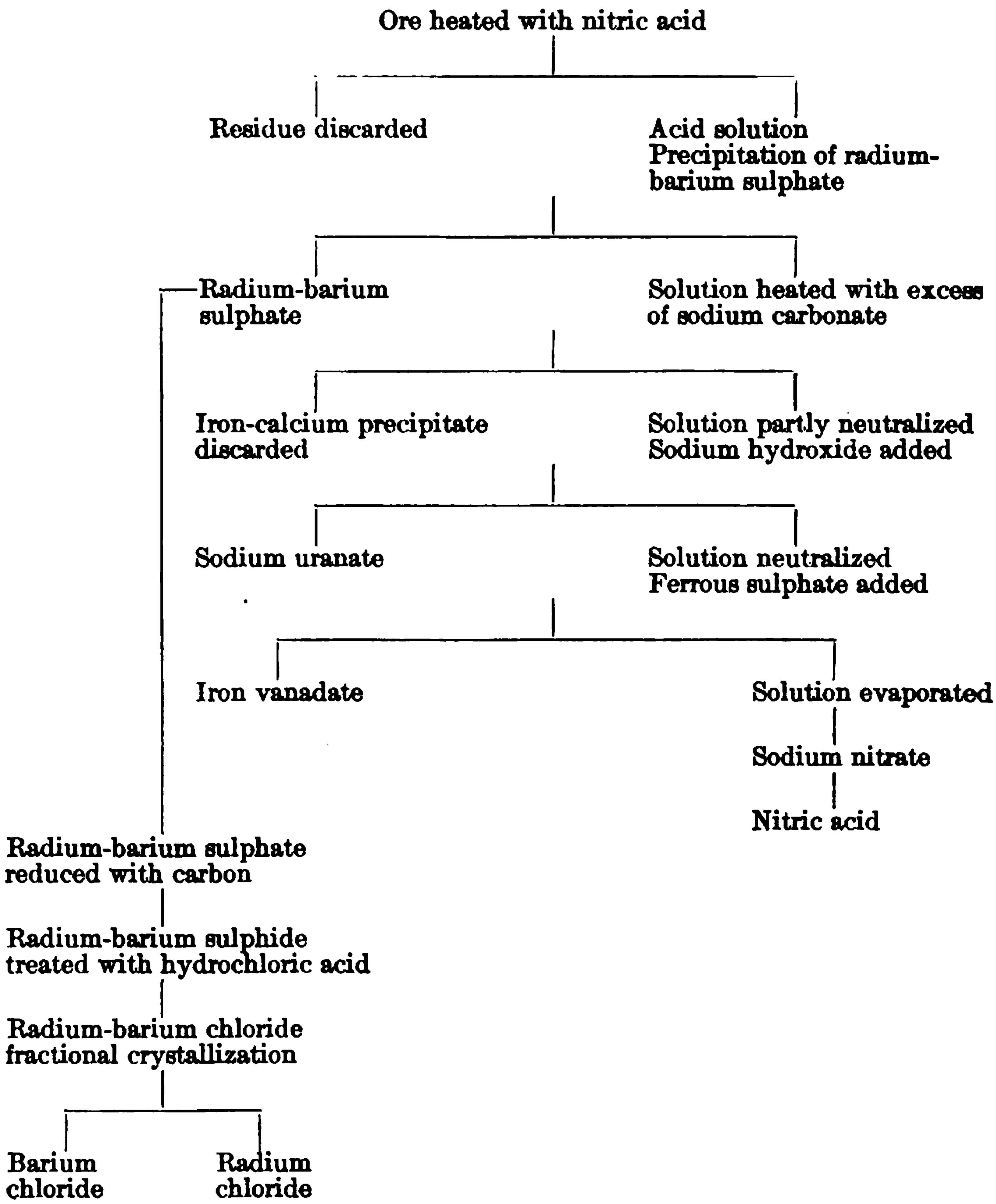
From the data already presented it can be readily seen that each particular type of radium-bearing ore has to receive a more or less different treatment, depending upon the other constituents of the ore. This process has been tried only with American carnotite, and may not be adapted to other radium-bearing ores, as its efficiency in connection with other ores has not yet been fully determined. The process is, however, applicable to the treatment of carnotite ores obtained in Colorado and Utah.

CHAPTER III. BUREAU OF MINES METHOD FOR TREATING CARNOTITE.

OUTLINE OF BUREAU OF MINES METHOD.

The method of treating carnotite ore used by the Bureau of Mines is outlined in this chapter. The process is clearly indicated in the diagram following:

Diagram of steps in Bureau of Mines method of radium extraction.



LEACHING, FILTERING, AND WASHING ORE.

In the method used by the Bureau of Mines, the ore is ground to 20 mesh and is leached with strong hot nitric acid in acid-proof earthenware pots. The amount of acid used is 121 pounds of 100 per cent nitric acid to 500 pounds of ore, the acid being diluted to 38 per cent strength. However, the strength may be varied somewhat, ores high in sulphates requiring stronger acid. The acid is brought near the boiling point by live steam, which has been passed through a baffle in order to eliminate any impurities from the steam. The ore is then run in, being stirred with a wooden paddle during the process. The acid is heated for 15 minutes longer, with occasional stirring, and the acid is then run into an earthenware vacuum filter, asbestos filter cloths being used. As much as possible of the sand is held back in the pot by means of a long wooden plug manipulated by hand, and this sand is given an acid wash with acid about one-third the strength of that used for the first leaching. This sand is then dumped on the filter, and receives two washings with hot distilled water.

PRECIPITATION.

The whole work from start to finish, leaching, filtering, and washing, can usually be done in about seven hours. The residue is thrown on the dump and the filtrate is run through earthenware pipes into a large precipitating tank made of California redwood, where it is diluted by the addition of water. This solution is stirred, and sodium hydroxide is run in slowly, with the object of reaching as nearly as possible the neutral point without forming a permanent precipitate. If too much alkali is added, both iron and vanadium are precipitated, and contaminate the radium-barium sulphate; on the other hand, if not enough sodium hydroxide is added, the acidity remains too high and the solvent action of the nitric acid on the radium-barium sulphate is not sufficiently decreased. A little practice enables the operator to get the right point by visual observation without titrating, titration not being possible because of the large amount of dissolved material that would be precipitated by an alkali.

A solution of barium chloride is then added, usually in the proportion of about 2 pounds of barium chloride to 1 ton of ore, and after the liquid has been thoroughly stirred, sulphuric acid is slowly added, as stirring continues. Fifteen pounds of sulphuric acid to 1 ton of ore is the preferable quantity. The stirring is continued for one hour, when the whole solution, containing the barium-sulphate precipitate, is elevated to a conical settling tank by means of a centrifugal pump having parts that come in contact with the liquid made of duriron. The solution is then allowed to settle in the conical tank for a few days. In the original plant this period was three days; in the second

plant it was increased to four. By means of a floating siphon, the clear solution is decanted into a tank containing an excess of boiling sodium carbonate, where the iron, calcium, and most of the aluminium are precipitated, and the uranium and vanadium go into solution as the double carbonate of uranium and sodium, and as sodium vanadate. The solution must be boiled for at least three hours after all the acid solution has been run in, as otherwise there is considerable loss both in uranium and vanadium, which remain with the iron precipitate.

The radium-barium sulphates and the associated liquor are run onto an earthenware suction filter, filtered, washed, and finally treated with a dilute solution of sodium hydroxide in order to remove the last traces of free acid. The filtrate is run into the carbonate tank with the rest of the acid liquor. The radium-barium sulphates are placed in iron pans and dried in a hot-air oven.

TREATMENT OF SODIUM CARBONATE SOLUTION.

The sodium carbonate solution, carrying the uranium and vanadium, is nearly neutralized with nitric acid, the solution being constantly stirred by means of compressed air; then sodium hydroxide is added to the boiling solution until there is a complete precipitation of sodium uranate. No attempt has been made to control definitely the color of this precipitate, as the sodium uranate has always been converted into oxide for final sale. In addition, the sodium uranate has always carried vanadium, as precipitation of the uranate in the presence of vanadium has so far always caused the precipitate to contain at least 7 or 8 per cent of V_2O_5. Consequently, it is generally necessary to remove the vanadium from the sodium uranate before it can be sold as such. Some early experimental work showed that redissolving with sulphuric acid and reprecipitating with sodium hydroxide would be necessary three or four times before the vanadium could be reduced to less than 1 per cent. This method of refining, therefore, could not be used commercially, and it became necessary to find some other cheaper and more efficient process. Such a process is described in subsequent pages.

TREATMENT OF VANADIUM SOLUTION.

The hot solution from the sodium uranate is completely neutralized with nitric acid, air being blown into the liquid in order to eliminate the carbon dioxide. Ferrous sulphate is then added, the liquid being continually agitated, and the precipitate of iron vanadate is filtered and washed.

The grade of the precipitate depends largely on the final acidity of the solution. If the solution is slightly acid, a high-grade precipi-

tate carrying 40 or 42 per cent V_2O_5 may be obtained. On the other hand, some of the vanadium is not precipitated and is therefore lost. If the solution is perfectly neutral after the addition of the ferrous sulphate, a lower grade product is obtained, carrying 30 to 33 per cent V_2O_5, but all of the vanadium is precipitated. It is advisable not to boil the solution after the addition of the ferrous sulphate, although the solution should be hot at the time of this addition.

TREATMENT OF SODIUM NITRATE SOLUTION.

The filtrate from the iron vanadate is almost wholly a solution of sodium nitrate, the main impurity being a moderate amount of sodium sulphate. The solution is evaporated in iron tanks heated by steam under pressure. Air from a compressor is blown into the solution and evaporation is rapid. As soon as the solution is concentrated enough it is run into steel crystallizing pans where it crystallizes. After draining on draining boards the crystals are collected without further treatment and employed to make fresh nitric acid for use in the plant. As the losses of sodium nitrate are not great, the actual results have been to reduce the cost of the nitric acid below the purchase price of hydrochloric acid of the same acidity.

RECOVERIES MADE BY BUREAU'S PROCESS.

The extraction and recovery of the radium have been excellent even from the start, an extraction of more than 90 per cent having been attained with many carloads of ore. The extraction of the uranium is also practically complete, but a considerable part of the vanadium is left in the ore. In fact, the presence of much vanadium in the ore is a disadvantage, as in the leaching pot the vanadium has a tendency to separate out as vanadic acid, which retards filtration considerably, and tends to reduce the extraction of the radium. Recovery of a few per cent more of the radium content more than compensates for a loss of 50 per cent of the vanadium, and if both can not be recovered, preference must be given the radium. The vanadium in the carnotite goes into solution readily, but roscoelite and other vanadium minerals present are decomposed with much difficulty, and it is not possible on a commercial scale to extract all of the vanadium in the ore, although complete extraction can be attained in the laboratory if enough acid is used.

GENERAL DESCRIPTION OF DENVER PLANT AND EQUIPMENT.

When the plant of the National Radium Institute was projected, it was to be entirely experimental. At the same time, plans were made to build and equip it so that if the experimental work were

successful, operation on a commercial basis would be possible by enlarging the plant or by using it as built. Therefore, it was necessary to design, erect, and equip a building that could be used permanently on a commercial scale, and yet to keep the cost down so that if the experimental work were a failure the loss would be as low as possible. For this reason, some equipment was not put into the plant that would have made the original work a little more efficient if the initial investment had been a little larger.

After the experimental work had proved successful, the officials of the National Radium Institute decided that they wished the radium delivered more rapidly than was possible with the original small plant (Pl. I, *A*). Consequently, an additional plant (Pl. I, *B*) was built adjoining the first one. Both were equipped as a separate unit so that they could be run separately or at the same time. For convenience of description, these plants are mentioned in this report as the "first" plant and the "second" plant.

Refining of the radium concentrate is done in a separate building, usually termed the "sulphate building."

During the first six months of work, nitric acid was purchased, but the sodium nitrate was recovered and stored. In December, 1914, and January, 1915, a nitric acid plant was erected, and since then the institute has made its own nitric acid, from the recovered sodium nitrate.

All the buildings of the plant proper are of frame construction, with an outer covering of galvanized iron painted on the inside with one coat of graphite paint to protect it from acid fumes. The roofs of the buildings are of elaterite.

The original or "first" plant (Pl. I, *A*) is 80 feet long, 45 feet wide, 21 feet high on the south, sloping to 18 feet on the north. Adjoining this is a storeroom (shown at extreme left of Plate I, *A*) with doors opening on to a switch that runs past the plant, connecting with both the Denver & Rio Grande and the Colorado & Southern Railroads. The original storeroom was only one story high, but later an additional story was added, as well as a small grinding and sampling plant, which was placed in a room built into one corner of the storeroom.

The "second" plant is 130 feet long, 30 feet wide, and 24 feet high on the north, sloping to 21 feet on the south. The old boiler room was turned into a furnace room for the refining of the uranium, and for the preliminary treatment of the radium-barium sulphates. A new boiler room to serve both plants was erected at the same time as the second plant. It is shown at the right of Plate I, *B*; a plan and sections of this building are shown in Plate II.

A. GENERAL VIEW OF PLANT OF NATIONAL RADIUM INSTITUTE FROM NORTHWEST SHOWING ORIGINAL OR "FIRST" PLANT ADJOINING BRICK BUILDING ON LEFT.

B. VIEW OF PLANT FROM SOUTH, SHOWING NEW OR "SECOND" PLANT.

DESCRIPTION OF FIRST PLANT.

The floor plan of the original or "first" plant is shown in Plate III and the plan of the upper landings on the south side in Plate IV, *A*. The ore and acid are carried to the upper landing by means of the elevator (see Pl. III), the acid being transported in carboys. The handling of the acid in the carboys has been fairly satisfactory, but an improvement could be made by conveying the acid in glass pipe lines to the leaching pots (*27a*, *27b*, etc., Pl. IV, *A*), which would partly eliminate the handling of the acid by hand. This was one of the changes that it was not thought necessary to make under the conditions.

CONSTRUCTION AND OPERATION OF LEACHING POTS.

Each of the earthenware pots in which the leaching is done has a capacity of 107 gallons. Originally each was set in an iron container holding oil that was heated with a steam coil. It was thought that some external method of heating would be necessary in order to quickly raise the temperature of the acid to the desired temperature and to maintain it at this temperature during the period of heating. The results were not satisfactory, as many of the leaching vessels cracked, probably from somewhat uneven heating and the strains to which they were subjected by the packing. During the early stages of the work, in addition to the heating by the oil bath, live steam was run into the acid through glass tubes connected with a pipe leading from a large baffle (*20*, Pls. III and V) that served to eliminate, as much as possible, impurities from the steam. Experience showed that the acid could be heated rapidly enough with live steam alone, and therefore the oil bath was discarded and sand was substituted. The results obtained were an improvement, not so many of the leaching vessels breaking, and the acid being heated at a satisfactory rate. With a view to getting still better results, cement jackets were finally tried, and two or three experimental pots were set in concrete. The results were so good that sand packing has been discarded. The breakage has been small and the cement jackets have proved satisfactory in every way.

The leaching pots are partly covered with wooden covers, each of which has a wooden flue leading to a main flue that goes through the roof (see Pl. IX, *B*, p. 46). A small steam pipe is placed in the main flue so that a jet of steam can be admitted when desired. By this means, practically all of the fumes from the acid can be removed without any trouble. The wooden covers of the pots and the wooden flues are coated with graphite paint, which makes a good protective agent. Stirring is done by hand with wooden paddles (Pl. IX, *B*),

as it is almost impossible to install mechanical stirrers that would be satisfactory under the conditions.

During the period of leaching the spouts of the leaching pots are closed with rubber stoppers fastened to cast-iron plates attached by sleeves and set screws to horizontal rods, which are turned by levers inserted through the floor of the upper landing (fig. 1). The stoppers can be removed from the spouts or replaced by means of these levers, which are worked from the upper landing, so that all danger from the splashing of hot acid during the dumping of a pot is eliminated. Such splashing, however, can be reduced to a minimum by inserting a wooden plug at the end of a handle in the upper part of the outlet of the pot before the stopper is removed. When the stopper is taken out and the acid begins to run from the pot, the plug is sucked into the opening. Raising the plug when necessary permits the acid to flow onto the suction filter below at any desired rate.

The acid wash is contained in a small wooden tank (*36*, Pl. IV, *B*). From this tank it runs by gravity through an ordinary acid-proof rubber hose to the leaching pots for washing the ore.

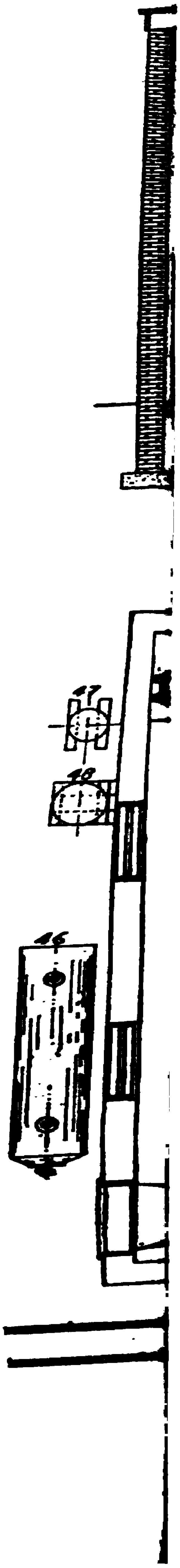
CONSTRUCTION AND OPERATION OF SUCTION FILTERS.

Plate IV, *B*, showing a section of the first plant, indicates the arrangement of the leaching pots (*27a*, *27b*, etc.) and of the suction filters (*28a*, *28b*, etc.) below. These filters were made by the German-American Stoneware Works Co., and are of the "Edda" type (see Pl. IX, *C*, and fig. 1). The upper part of each filter has a capacity of 105½ gallons, as has the lower part. Each bottom vent has a ground-in bibcock, and the upper opening is connected by means of about 6 feet of ½-inch pressure rubber tubing with a pipe running to the vacuum pump. This pipe first connects with a baffle (*47*, Pls. VI and VII), which protects the pump itself. This baffle is filled with vitrified brick, and a strong solution of sodium hydroxide constantly circulates through it.

Each filter is set on a small truck (see Pl. IX, *C*) running on rails so that when the residue is to be removed from the filter the truck can be pushed or pulled from underneath the leaching vessels. The filtering medium consists of an asbestos filter cloth covered with 1 inch of coarse sand over which there are several strips of wood. These strips are held down by three or four bricks suitably placed.

The suction filters have lasted even better than the leaching pots. Before the hot acid is dumped, a small piece of steam hose connected with a steam line is pushed up through the bibcock of each filter so that the lower part is slowly steamed and the temperature raised gradually. Another similar piece of hose, connecting with a second steam line, is pushed down into the upper part of each filter and the

BUL



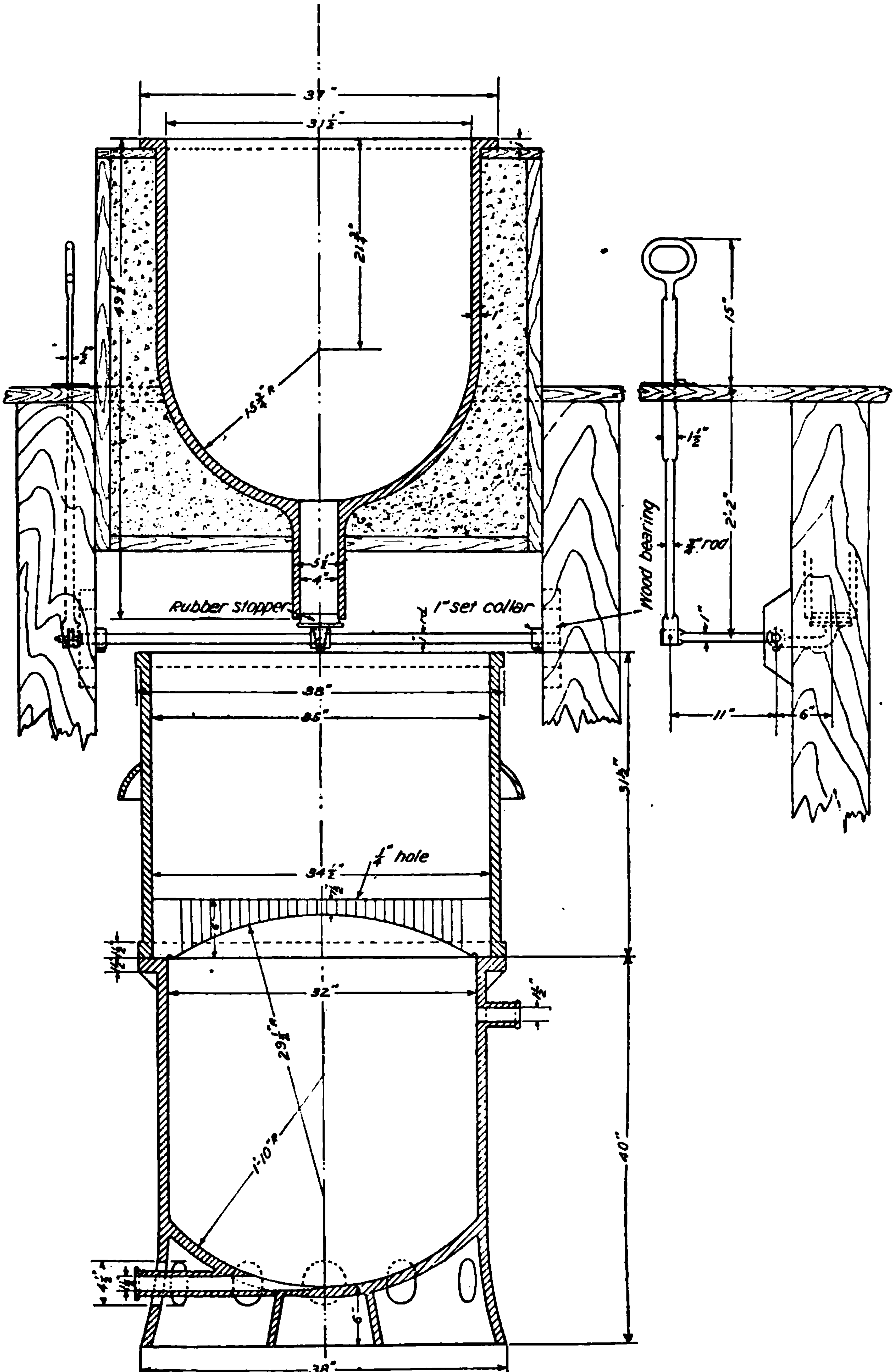


FIGURE 1.—Sections showing leaching pot, filter, and details of leaching-pot stopper.

whole covered by any suitable material as oilcloth or sacking. In this manner, the upper part of the filter is also steamed and the temperature gradually raised before the hot acid is dumped. With these precautions, few breaks have occurred. The acid from the suction filters flows through a horizontal, acid-proof earthenware line which dips slightly toward the radium precipitating tank, and has an opening below the bibcock of each suction filter.

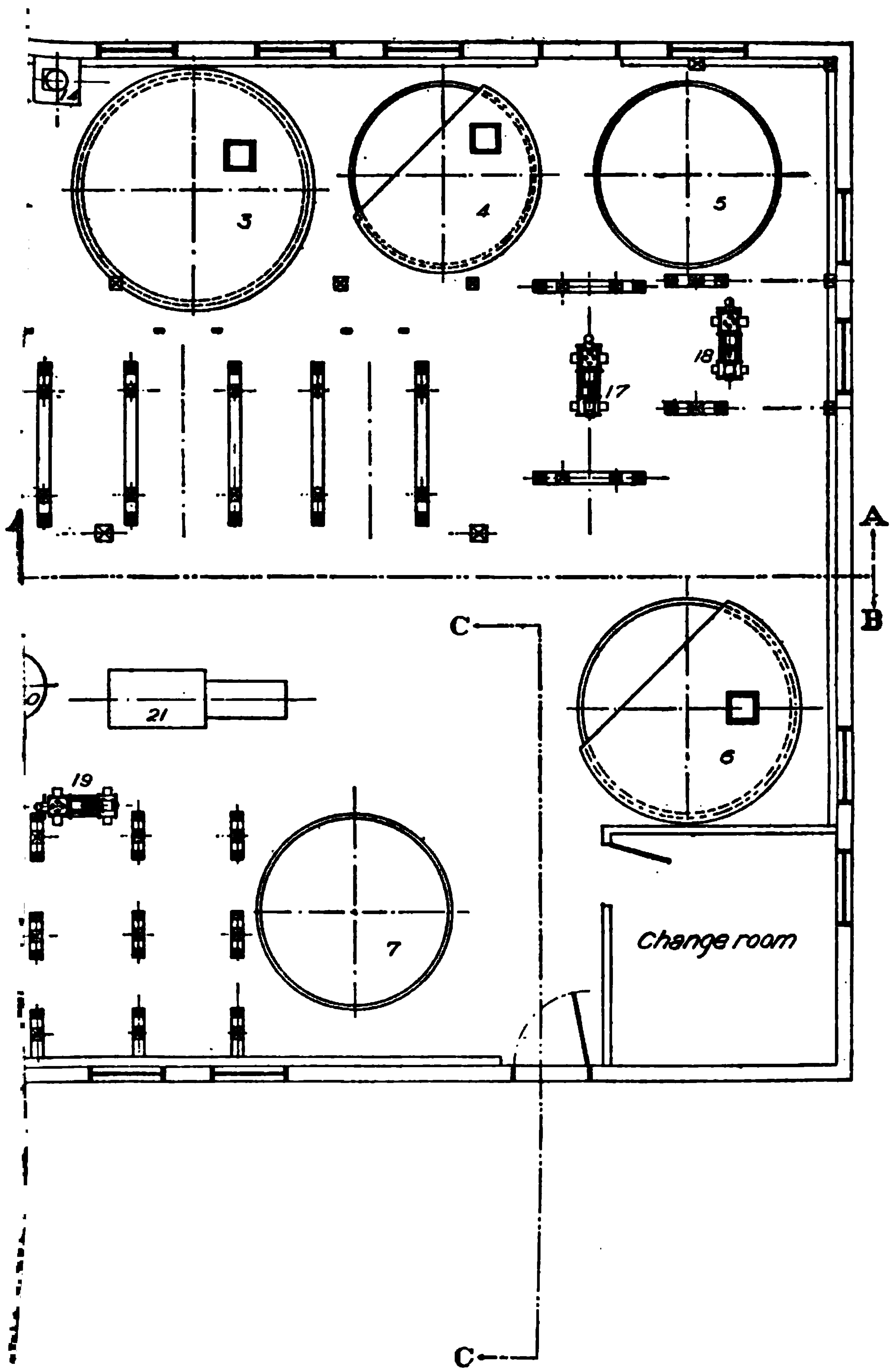
PRECIPITATING TANKS.

The precipitating tanks (Pls. III and VII) are all of 2-inch California redwood, which has satisfactorily withstood the action of acids and of weak alkalies. The radium precipitating tank in the first plant, at the time of writing (September, 1915) has been in use 15 months, and is still in fair condition, although it has been subjected daily to the action of warm dilute nitric acid. Those tanks in which solutions are boiled have a wooden cover, with a wooden flue 12 inches square that extends up through the roof, so that little steam escapes within the plant.

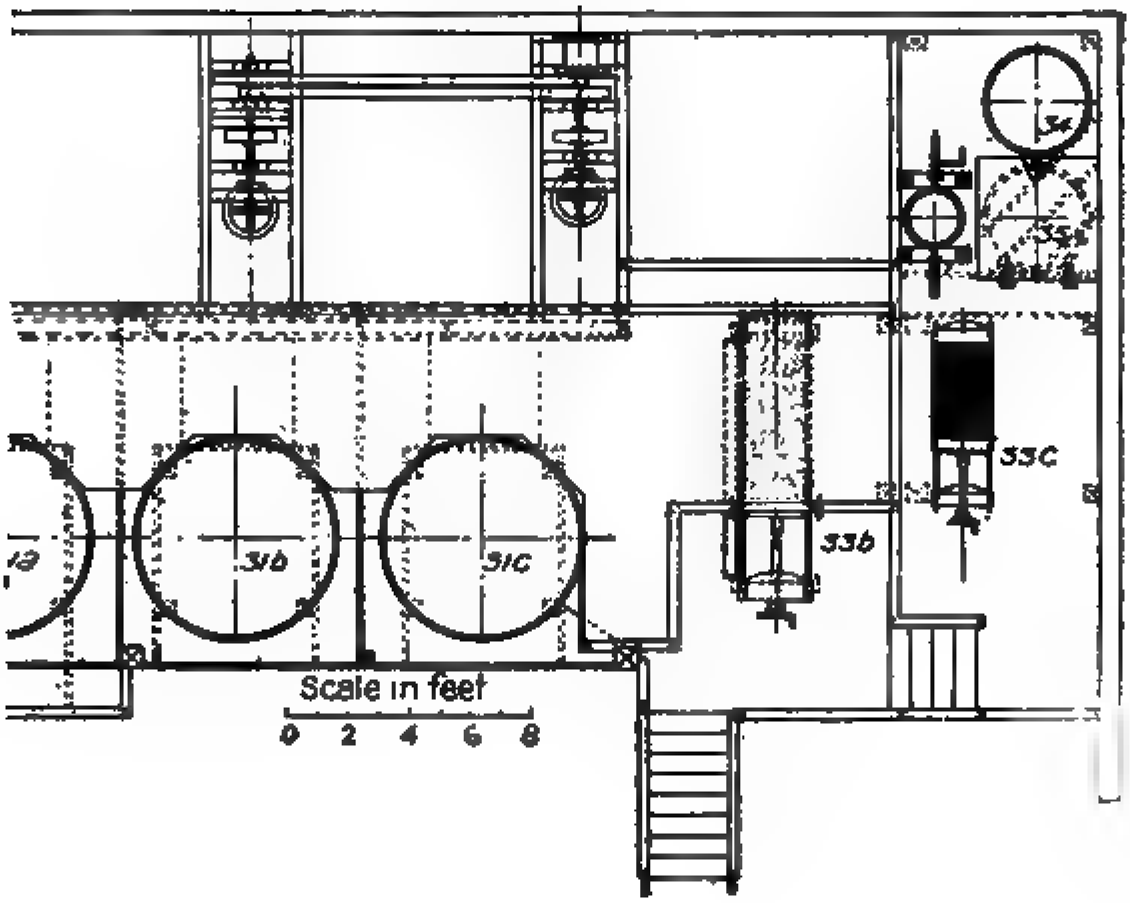
The bottom part of each settling tank is conical, although the vertical staves, which constitute the outside of the tanks, extend the entire length. In other words, these tanks are ordinary 8½ by 5 redwood tanks with a cone inserted in the bottom, and can be supported from below instead of being suspended.

There was considerable difficulty in getting these tanks tight. Ordinarily, wooden tanks are built to hold water and any small leaks are closed by the swelling of the wood. When acid solutions are poured into wooden tanks, the wood tends to contract, rather than to swell, so that acid tanks must be set up differently. The staves must be properly machined and all joints made with special care, extra lugs must be used, and the use between the staves of a rubber cement is advisable. A tank must be absolutely tight when liquid is first put in. However, if these precautions are taken, a tank can be kept tight without serious trouble, an occasional tightening of the lugs or a little calking being all that is required. Oval holes were cut in the lower part of the tanks large enough to allow a workman to crawl in and thus get at the lugs holding the conical part of the tank (see Pl. X, *A*, p. 50). The lugs and, indeed, the whole exterior of the tanks should be kept covered with graphite paint. The other tanks were put up with a thin coating of white lead between the staves.

It has been found convenient to run the sodium carbonate into the tanks by bringing it on a small truck by means of the elevator (Pl. IV, A) to the third landing, and then letting it fall into the tank below through a wooden chute 8 inches square.



n tank; *8*, distilled-water tank; *9, 10*, storage
team trap; *15*, centrifugal pump; *16, 17, 18, 19*,



5, earthenware filter; 36, dilute-acid tank; 37,

33b, etc., suction filters; 39, tram car; 40, sodium
, 36, dilute-acid tank.

STEAM-HEATING DEVICES.

Each tank in which solutions are boiled contains a grid composed of 1½-inch Byers pipe with cast-iron fittings, the extent of heating surface in the different tanks depending, of course, on the object to be attained. In the larger tanks (10 to 12 feet in diameter) in which liquids are heated without being evaporated, 12 lengths 6 feet long have been found sufficient. In the nitrate storage tanks, where a certain amount of evaporation is desired, the heating surface is nearly twice that for the larger tanks. The steam flowing to these grids passes through a large baffle (*20*, Pls. V, VI, and VII) filled with vitrified brick, which eliminate impurities carried from the boiler. As the grids are also connected with suitable traps, a constant supply of distilled water is obtained and stored in a large iron tank (*8*, Pls. V, VI, and VII). From this tank distilled water is piped to the third landing to a point near the leaching pots, and by means of compressed air the water is elevated to this point whenever desired. As it is usually hot considerably less steam is needed in connection with leaching than if the water were cold. Exhaust steam is used for preliminary evaporation of the nitrate solutions.

FILTER PRESSES.

The filter presses used in the first plant are of the Shriver two-eyed "washing" type. The two larger presses (*33a*, *33b*, Pl. IV) contain 30 plates, each 24 by 24 inches. The smaller presses (*33c*, Pl. IV, and *33d*, Pl. V) contain 21 plates each, 18 by 18 inches. The iron precipitate press (*33a*, Pl. IV) is filled twice during the day's run, so that if one desired to empty this press only once a day, its capacity would have to be doubled. The small presses take care of the uranium and vanadium at one operation. Besides being connected with the tanks, the presses are piped for both water and compressed air. The pumps (*16* to *19*, Pl. III) connected with these presses are of the brass-lined, double-acting Worthington type.

ACID CONVEYORS.

The acid from the radium precipitating tank is elevated to the settling tanks by means of a centrifugal pump through 2-inch acid rubber hose. The parts of the pump that come in contact with the liquid are made of duriron, and have resisted the acid excellently, as has the rubber hose, which is lined with pure gum. The acid is transferred to any desired settling tank by running it through the hose into a wooden flume connecting the different tanks.

FLOATING SIPHON.

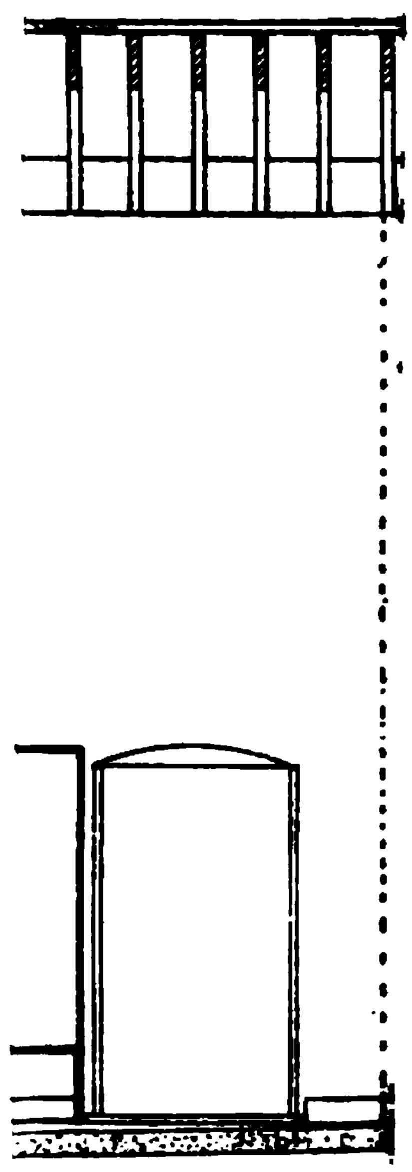
The floating siphon (Pl. X, *B*, p. 50) consists of a square wooden frame through the center of which a 1½-inch acid hose extends about 18 inches, or any other desired length. The frame is built so that it settles in the tank until at a certain point it rests on the sloping sides of the cone. The length of hose going through the frame is long enough to siphon off the clear liquid above but not to disturb the radium-barium sulphate precipitate. The other end of the hose is connected with a 1½-inch piece of glass tubing 5 feet long, from which another piece of the same type of hose runs into the tank. An earthenware stopcock is inserted at a convenient point near the glass tube, and is held by a chock so that the cock can be turned but can not be forced out by the pressure of the liquid. In this way the flow of the liquid can easily be controlled, and at the same time the liquid can be examined as it runs through the glass tube. As long as it is not turbid it is free from radium-barium sulphate precipitate. If the workmen, in occasional inspection of the tube, note any turbidity they can shut off the siphon and thus eliminate radium losses. In running the acid liquid into the carbonate tank, any convenient form of spreader can be used to prevent much acid striking the carbonate solution at any one point and thus reducing too much the alkalinity of the liquid at that point.

MISCELLANEOUS FEATURES.

After the clear liquid has been siphoned off, the radium-barium sulphate with the remaining liquor is introduced onto earthenware suction filters, which are placed below the settling tanks. As mentioned previously, these filters are also of the Edda type, the upper and lower parts each holding 52¾ gallons of liquid. They are placed on small trucks running on tracks, so that they can be pulled from underneath the settling tanks and the radium-barium sulphate easily removed (Pl. X, *A*).

A compressor delivering about 300 cubic feet per minute was originally situated on the north side of the plant (*21*, Pl. III). When the second plant was built, however, this compressor was removed and a larger one (*55*, Pl. II) was placed in the boiler house and now serves both plants.

The mechanical stirrers in the different tanks are belt driven, one motor driving the stirrers in tanks *1* and *2* (Pl. III, and Pl. IV, *B*), and another one driving those in tanks *3* and *4*. The liquids in tanks *5*, *6*, and *7* are agitated by compressed air, as the weight of the precipitates is not large and the time of agitation is not long. The stirrers in tanks *1* and *2* are of wood and have two blades revolving 8 inches and 18 inches above the bottom of the tank, and the stirrers

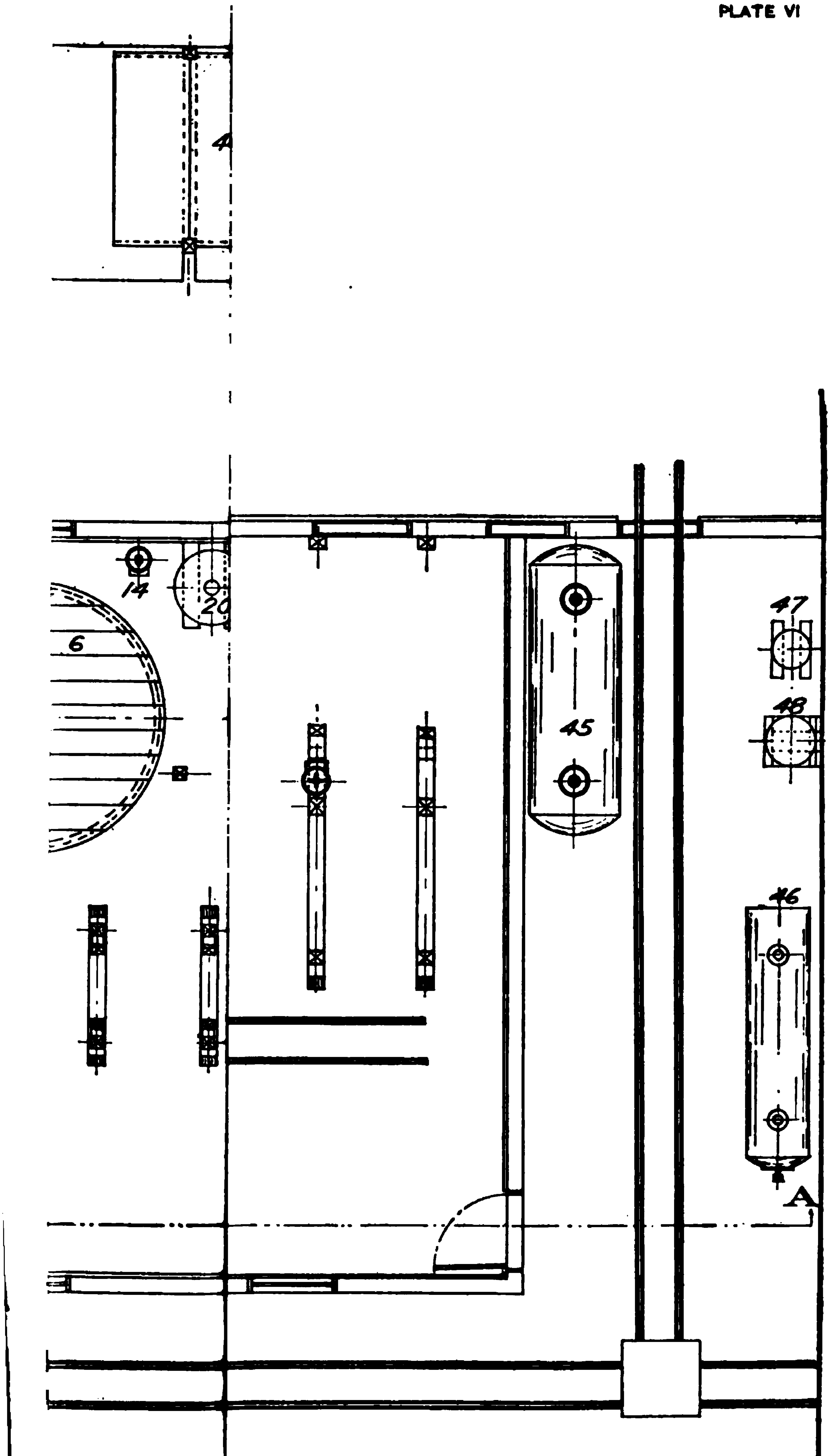


e; 22, steam drier; 23

PLA

k; 7, v
ɔdium

PLATE VI



PLAN OF NEW

k; 7, vanadium tan
ɔdium nitrate evap

in tanks *3* and *4* are of iron, and each has one blade revolving just below the heating coil and a second blade about 6 inches above the heating coil.

DESCRIPTION OF SECOND PLANT.

The general arrangement of the second plant is similar to that of the first, but as the building had to be of a different shape, some changes were made, as well as some improvements.

Instead of using Shriver presses, three 25-inch, lever-operated, clam-shell Sweetland filter presses (Pl. VIII) were installed. Each one of these has 130 square feet of filtering surface. One takes care of the iron precipitate from the sodium carbonate tank, the other handles the uranium, and the third the vanadium. As the total capacity of the last two presses is larger than is required by the one plant, the uranium and vanadium from both plants are now filtered through these presses, the solutions carrying the precipitated uranium and vanadium in the old plant being pumped into the corresponding tanks in the new. These presses are piped for water, compressed air (on both sides of the cloths), vacuum, and steam. It has been found difficult to get the cakes from the presses sufficiently dry to be dumped easily without the use of compressed air.

The filtrates from the iron and uranium precipitates go directly from the pumps into elevated tanks (*51*, Pl. VII). From these the iron filtrates are run by gravity into the uranium-precipitating tank (*6*, Pls. III, VI, and VII), and the uranium filtrates into the vanadium tanks (*7*, Pls. III, VI, and VII, and *39*, Pl. V).

The stirrers in the new plant are driven by worm gears instead of belts. The stirrer in the radium-precipitating tank under load makes 44 revolutions per minute and that in the sodium carbonate tank makes 16 revolutions per minute. Each stirrer is driven by a 2-horsepower motor. The duriron centrifugal pump used for elevating the liquid from the radium-precipitating tank is the same size as was used in the first plant; it is run at 2,000 revolutions per minute and is belt-driven from a 5-horsepower motor.

The sodium hydroxide, which is used for partial neutralization of the acid in the radium precipitating tank, and also for precipitating the uranium, is made up in a steel tank (*49*, Pl. VII), placed on an exterior platform connecting the new plant and the boiler house. The sodium hydroxide in the iron drum is broken with a sledge hammer, and dumped into this tank, the solution being agitated with compressed air. It is advisable to use comparatively little hot water until all of the sodium hydroxide has been dissolved and the solution has cooled somewhat, when it can be diluted. The solution is then run by gravity through a pipe into a large steel storage tank (*45*, Pls. VI and VII), from which it can be transferred by means of

compressed air to the measuring tanks (*30*, Pls. IV and VII), situated on the upper landing in each plant. By means of a float attached to a cord terminating in a weight, which rises and falls over a graduated scale, the amount of liquid taken from these tanks can be determined. From each of these measuring tanks pipes run to the radium precipitating tank and to the uranium tank in each plant.

The system for obtaining distilled water for the new plant is the same as that in the old, the distilled water reservoir (*8*, Pls. VI and VII) being correspondingly larger.

SULPHATE BUILDING AND EQUIPMENT.

The small building in which the radium barium sulphates receive preliminary treatment and fractionation (figs. 2 and 3) is situated

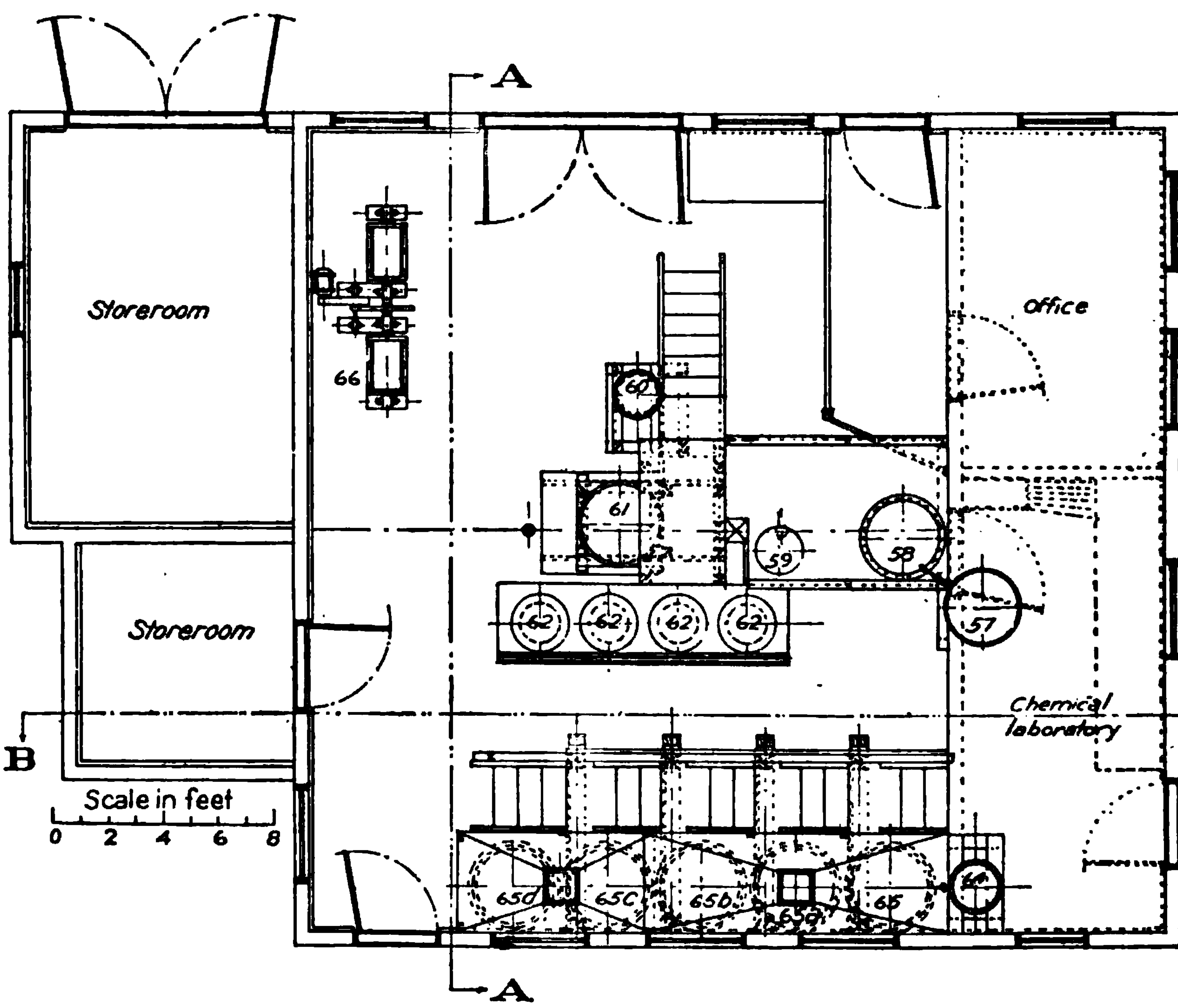


FIGURE 2.—Plan of sulphate building. *57*, condenser; *58*, distilled-water reservoir; *59*, baffle; *60*, experimental pressure apparatus; *61*, earthenware suction filter; *62*, silica-lined fractionating pot; *63*, dissolving pot; *64*, suction filter; *65*, *65a*, etc., steam-jacketed fractionating vessels; *66*, ball mills.

just behind the first plant. The arrangement as shown in the plan (fig. 2) is not exactly the arrangement that actually existed at the time of writing (September, 1915) because the silica-lined, acid-proof kettles ordered in France had not arrived; the arrangement used was more or less tentative. The plan shows the arrangement as it will be when completed.

In order to crystallize radium-barium salts from acid liquors, it is, of course, necessary to a certain extent to evaporate the solutions, in order to get the concentration necessary for proper crystallization.

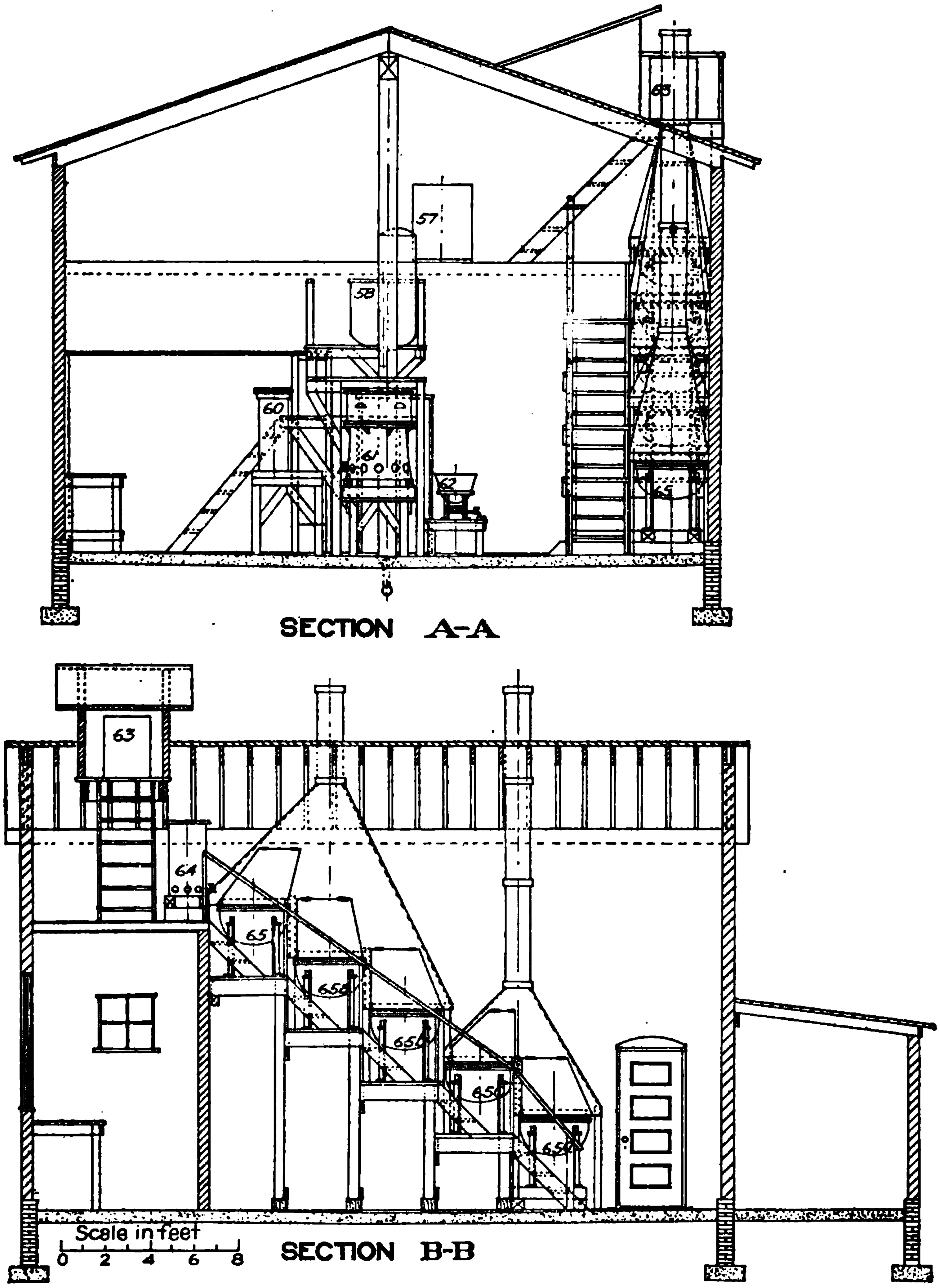


FIGURE 3.—Sections of sulphate building.

This might be done in earthenware vessels, provided a satisfactory and efficient method of heating could be obtained. Copper or silver steam coils might be used, but both of these metals are gradually attacked by concentrated hydrochloric acid, and in addition there

would be a tendency for crystals to form around the coils. Breaking these crystals away would tend to damage the coils, and it would be inconvenient to handle the crystals under such conditions. Any exterior means of heating, such as steam, boiling water, or hot oil, would involve too great a risk of breakage of containers and loss of valuable solutions. Large silica or porcelain basins have been used, and when the table below them is covered with sheet copper bent up at the sides and ends and brazed, so as to make a shallow vessel, such dishes can be used with reasonable safety. Their capacity, however, is too small for handling the quantities of material required at the plant of the National Radium Institute; therefore, it became necessary either to have larger vessels that would stand the action of boiling concentrated hydrochloric acid, or else to crystallize in neutral solutions. As crystallization in acid is much more rapid and efficient, the former method is much to be preferred.

A silica-lined, acid-proof ware made by Danto-Rogeat & Co., of Lyons, France, is exceedingly satisfactory. No other ware obtained either in this or any other country gave satisfactory results. In fact, most of the so-called "acid-proof ware" failed absolutely to withstand the acid. The institute was able to procure a number of small-size vessels from Danto-Rogeat & Co., and also, through the courtesy of the Welsbach Co., of Gloucester, N. J., to obtain one 250-liter steam evaporator made by the same concern. Owing to the European war, Danto-Rogeat & Co. were unable to supply any large-size vessels, as they had none in stock and their factory was closed. In September, 1915, they were about to start operations again, and an order for a number of steam evaporators has been placed for the purpose of equipping the plant as indicated in figures 2 and 3.

Distilled water is obtained for this part of the work by passing live steam through baffle *59* (fig. 2), and thence through a 1-inch block-tin coil contained in condenser *57*, which consists of an ordinary wooden tank with the necessary inlet and outlet for a water flow. The block-tin pipe runs into earthenware distilled-water reservoir *58*, which has a capacity of 107 gallons, and the water is piped to any part of the building desired.

In the sulphate building are the office and a small chemical laboratory, which is used mainly for qualitative determinations, titration of acids, etc., or any chemical work in which a quick result is desired. The main chemical control work and radium measurements, as well as the final refining of the radium, has been done in the laboratories of the Bureau of Mines. There is also a small storeroom, built of reinforced concrete with a steel door, in which the radium-barium sulphates and other valuable material can be stored, so as to eliminate fire risk. The building also contains a small double ball mill,

VIEW IN NEW PLANT SHOWING FILTER PRESSES.

66, motor driven, for grinding the sulphates and mixing them with charcoal for later reduction in an oil furnace.

POWER HOUSE.

The 90-horsepower Kewanee locomotive-type boiler, used for the first plant alone, was purchased when the plant was on the experimental basis. When it was decided to build the second plant and continue work during the full period covered by the agreement with the Bureau of Mines, an additional 150-horsepower tubular Kewanee boiler was installed. The boiler house (Pl. II) is of brick, with concrete floor, a brick-and-iron partition separating the boilers from the compressor and pumps. The coal bin will readily hold about 2½ cars of coal.

The compressor (*55*, Pl. II) is 16 by 18 inches, capable of 140 revolutions per minute, and delivers about 650 cubic feet per minute at 40 pounds. The main vacuum pump *54*, a 10 by 20 inch Rotrex pump, belt driven, is connected with the suction filters used for leaching in both plants, and, with all pots connected, will give a vacuum of 20 to 22 inches of mercury. It is protected by means of a baffle, *47*, filled with brick, over which a strong solution of caustic soda slowly circulates. The small pump *54a* was the one originally used in the first plant, and was transferred to the power-house when the second plant was built. It is a No. 3 Nash vacuum pump; is chain driven, and will give a vacuum of 15 to 17 inches of mercury. It is connected with the suction filters under the settling tanks and also with the sulphate building, and is protected by a baffle (*48*, Pls. VI and VII).

Between the sulphate building and the extension of the new plant, is a small building with concrete floor and concrete walls for the storage of sodium nitrate. This building is connected by means of a belt elevator (Pl. XII, p. 60) with the bin in which the sodium nitrate is bedded and sampled, and from thence the nitrate is taken to the stills.

EQUIPMENT FOR SODIUM NITRATE RECOVERY.

The sodium nitrate is pumped from the storage tanks (*11* and *12*, Pls. III, VI, and VII) in each plant by means of a Worthington pump, 4½ by 2¾ by 4 inches, into the two evaporators (*43*, Pl. VI). These are made of ⅜-inch steel, one being welded and the other riveted, each being 6 feet by 5 feet by 24 inches, with a slope toward the center. Steam is used for evaporating, each tank containing a grid made of 1½-inch Byers pipe. Compressed air is used to hasten evaporation. The tanks are elevated so that the solutions can run by gravity into the crystallizing pans (*44*, Pl. VI, and Pl. XI, *A*),

which are made of ⅛-inch steel. Each pan is 10 feet long by 5 feet wide, 8½ inches deep at one end and 12 inches at the other, sloping toward the deep end so that it will drain readily. The pans are protected by a wooden roof covered with elaterite.

FURNACE ROOM.

When the second plant was built, the original boiler house was converted into a furnace room. This is equipped with three Case oil furnaces (*23*, *24*, *25*, Pl. III), the blast being furnished by fans driven by 1-horsepower motors. The oil is supplied under pressure from a tank outside the building. The furnaces were specially built, two of them (*24* and *25*) holding cast-steel pots 18 inches high by 16½ inches wide and three-fourths of an inch thick (Pl. XI, *B*, p. 58). The other (*23*), a little larger in size, holds three No. 100 graphite crucibles, and is used in the reduction of the radium-barium sulphate. In the same room is a small ball mill (*26*, Pl. III), 2 feet 10 inches by 2 feet 6 inches, in which the sodium uranate is ground, either alone or with any other material that is desired in its treatment.

DETAILED DESCRIPTION OF OPERATIONS.

LEACHING.

HANDLING OF NITRIC ACID.

The nitric acid is handled in carboys, and was originally weighed on scales at the foot of the elevator close to the sampling room. Since the completion of the acid plant (Pls. XII and IX, *A*) it has been weighed as drawn from the storage pots. With the weighed ore it is carried by means of the elevator to the upper landing of the first plant, from which there is a connection to the upper landing of the new plant, so that the same elevator serves both plants.

The spout of each leaching pot (fig. 1) is filled with freshly washed sand in order to protect the rubber stopper, and the acid is then dumped from the carboys into the leaching pots (Pl. IX, *B*), each of which contains sufficient distilled water so that when the pot has been filled it contains 121 pounds of 100 per cent nitric acid diluted to 38 per cent strength. Each day the foreman in charge of this work receives a slip indicating the weight of acid to be weighed for each pot and the volume of distilled water to be added.

HEATING NITRIC ACID.

Live steam is run into the acid through a ½-inch glass tube connected by rubber tubing with a steam line. The steam in the line is passed through a baffle so as to eliminate impurities, and contains only traces of sulphates. As the distilled water is usually hot the

A. NITRIC ACID PLANT, FROM SOUTH.

B. BATTERY OF LEACHING POTS IN NEW PLANT.

C. SUCTION FILTERS IN NEW PLANT, SHOWING LEACHING POTS ABOVE.

time of heating is not long, but steam is run in until the temperature of the acid is at least 85° C.

The ore, ground to 20 mesh, is slowly added to each pot, the ore sacks resting on the wooden coverings (Pl. IX, *B*) on the pots, and the workmen using wooden paddles (Pl. IX, *B*) to stir the acid as the ore enters. Frequently calcium carbonate in the ore causes effervescence, and care has to be taken that the ore is not added so rapidly that frothing makes the acid overflow. Five workmen and a foreman can handle the 14 pots that are in the two plants and do all the necessary work connected with this part of the process. The heat of solution makes the temperature of the acid gradually rise during the addition of the ore, the temperature finally reaching 91° or 92° C. The addition of steam continues for 15 minutes after the last of the ore is in, the workmen stirring the different pots as rapidly as possible during this heating.

Nitric acid is not a very good solvent of vanadium, although it readily breaks up the vanadium minerals, even roscoelite. On the other hand, if too much vanadium is present, there is a tendency for vanadic acid to separate out in the acid as a reddish-brown precipitate. As this is rather gummy it retards filtration. If considerable vanadium separates in this manner, filtration may be so retarded that the acid cools down, causing a considerable loss of radium by reprecipitation. If there is little or no separation of vanadium in the acid, filtration takes place readily and quickly, and the extraction of the radium is thereby increased. Therefore, if the heating is continued with the object of decomposing the vanadium minerals other than carnotite, there is likely to be a loss of radium that will much more than compensate for the increased yield of vanadium. Consequently, it is much better not to continue the heating too long, but to make the extraction of the radium the main object. The uranium is practically all dissolved under the conditions mentioned.

USE OF HYDROCHLORIC ACID.

The addition of a small amount of hydrochloric acid to the nitric acid increases the solvent action on the radium. Therefore, if the nitric acid does not contain hydrochloric acid, enough hydrochloric acid is added to make the proportion of concentrated hydrochloric acid to 100 per cent nitric acid, about 2.5 per cent. In other words, to each pot, 9 pounds of 28 per cent hydrochloric acid, the strength of the acid available in Denver, is added. This hydrochloric acid, of course, appears as sodium chloride mixed with the sodium nitrate, and the nitric acid made from the sodium nitrate contains some hydrochloric acid.

When heating has continued long enough, a wooden plug on the end of a long handle is inserted into the pot and pushed through the ore until it almost completely stops the inside opening of the spout. By means of a lever (fig. 1) the rubber stopper is then removed from the lower part of the spout, and by raising the plug a little the acid is allowed to run onto the upper part of the suction filter (Pl. IX, *C*) below. In this manner, the flow of acid can be readily controlled, and at the same time a considerable part of the ore is retained in the leaching pot and receives the benefits of the acid wash. Thus, this method of handling the material has partly the effect of decantation.

USE OF ACID WASH.

An acid wash that has been prepared and heated in a small wooden tank, of which there is one in both plants (see *36*, Pls. IV and VII), is run into each pot, and the heating, by means of live steam, continues. The acid flows by gravity through an acid-proof rubber hose, and each pot receives an acid wash of about 170 pounds of 10 per cent nitric acid. When the acid leach on the suction filters has gone through, the acid wash, with the remainder of the ore, is dumped onto the filters in the manner described above.

WASHINGS WITH DISTILLED WATER.

When this material has completely filtered, the ore receives two washes of hot distilled water of 200 pounds each. Before the distilled water is added, however, the vacuum is broken on the filters and the liquid below, which consists of the acid and acid wash, runs from the bib cocks with which each filter is provided and flows through an earthenware pipe, which connects the different filters, to the radium precipitation tank (*1*, Pls. III, IV, VI, and VII). This tank contains 5 or 6 inches of water so that the hot acid is diluted enough not to damage the tank seriously. When the distilled water washes have filtered they are added in the same way to the acid liquor. Usually, the leaching process and filtration are complete between 2 and 3 o'clock in the afternoon. While the filtering is actually under way the workmen are weighing out the acid and ore for the next day's run, and transporting it from the first floor to the third, so that leaching on the next day can begin promptly. As soon as filtration is complete the residue, consisting mainly of silica, is shoveled from the upper parts of the filters into a small trolley, which runs along the platform on which the filters are placed, and hauled to the dump.

ADDITION OF SODIUM HYDROXIDE.

The acid solution in the radium precipitation tank (*1*, Pls. III, IV, VI, and VII), together with the wash water, is partly neutralized with sodium hydroxide, which is contained in small iron reservoir tanks

(*30*, Pls. IV and VII). The tanks are filled as needed through pipes from the main sodium hydroxide reservoir (*45*, Pls. VI and VII), the liquid being propelled by means of compressed air. Sodium hydroxide is added at first rapidly and then slowly, the solution being stirred all the time. After a certain amount of the sodium hydroxide has been added a greenish precipitate appears in the tank. As more alkali is added this precipitate gets heavy and finally tends to turn slightly brown. At this stage the addition of the alkali should stop. As a rule, about 70 pounds of sodium hydroxide is required per ton of ore treated, but no set rule as to quantity can be given, as the iron and vanadium content of the ore varies. The more iron and vanadium the ore contains, the less sodium hydroxide can be added without carrying the precipitation too far.

RADIUM PRECIPITATION.

About 2 pounds of barium chloride, in solution, per ton of ore treated is then poured into the tank. As a rule the barium chloride is that recovered in fractionation and contains small quantities of radium. After the solution has been stirred for five minutes in order that the barium chloride may be well mixed in, sulphuric acid is added, about 15 pounds of 100 per cent acid per ton of ore treated. The sulphuric acid not only precipitates barium sulphate, which carries down the radium, but also dissolves the small precipitate of iron and vanadium in the solution. Consequently the final precipitate obtained is a fairly clean one of radium-barium sulphate, containing only small traces of iron, vanadium, etc.

Should the addition of the sodium hydroxide be carried too far, the precipitate obtained is what is called a "retreat"; that is to say, it carries more iron and vanadium than is advisable. Under such conditions the wet sulphate is placed in an earthenware pot (*50*, Pl. VII) and concentrated sulphuric acid is added, the whole being thoroughly stirred. Water is then run in and the iron and vanadium readily dissolve in the hot liquor obtained, leaving the radium-barium sulphate as a clean white residue. The liquid and residue are run through a rubber hose to one of the filters (*32*, Pl. VII). Each filter is used only one day out of four for filtering the regular precipitate from the settling tanks (Pl. X and *31*, Pl. VII). The amount of sulphuric acid required for leaching a retreat varies according to the amount of iron and vanadium in the precipitate, but usually 25 to 35 pounds of 66° B. acid suffice. The acid filtrate obtained is used as part of the sulphuric acid required for precipitating the next batch of radium-barium sulphate in the precipitation tank (*1*, Pl. VII).

After the precipitation of the radium-barium sulphate, stirring is continued for one hour, when the whole solution is pumped into one

of the settling tanks (*31*, Pl. VII) through the centrifugal pump (*15*, Pls. III, VI, and VII). In the first plant the settling process occupied three days, as there were only three settling tanks; in the second plant the liquids are allowed to settle four days, there being four tanks. On the second day after the liquid has been run into the tank a solution containing 1 pound of barium chloride is poured in and thoroughly stirred into the upper part of the acid liquor by means of a paddle. The object is to carry down any small traces of radium that may have remained suspended or dissolved in the supernatant liquid.

PRECIPITATION PROCESSES.

IRON PRECIPITATION.

USE OF SODIUM CARBONATE.

When the settling is complete the clear liquid above the precipitate is siphoned off through a 1¼-inch acid rubber hose attached to a floating siphon. The acid liquor is run into tank *3* (Pls. III, VI, and VII), which contains a hot solution of sodium carbonate more than sufficient to neutralize the acid. The weight of sodium carbonate used depends to some extent on the character of the ore treated. The smaller the amount of uranium and vanadium in the ore, the smaller the excess of carbonate required. For ore running from 2.5 to 3 per cent U_3O_8,[a] 250 pounds of excess sodium carbonate is required per ton of ore treated. This figure does not take into consideration the addition of the sodium hydroxide in the radium precipitation tank, but is calculated on the original acidity of the acid used. Consequently if acid of the strength and in the quantity described under the section on leaching is used the actual weight of sodium carbonate used is about 650 pounds of soda ash per ton of ore. In reality the excess of sodium carbonate is larger than that indicated, owing to the fact that a part of the acid is neutralized by the sodium hydroxide added in tank *1*. A 2 per cent U_3O_8 ore would not need more than 200 pounds excess sodium carbonate per ton of ore unless it contained more than the usual amount of vanadium.

The acid liquor is siphoned into the hot sodium carbonate, the whole process taking about three hours. As long as the alkali is in excess, there is little chance of the solution boiling over; should the neutralization point be nearly reached, there is serious danger of such boiling. The liquid is heated during the addition of the acid, and maintaining the temperature near the boiling point during the entire period is important; otherwise the precipitate, which should be red, becomes brown or even bluish-brown, and carries considerable vana-

[a] The term "U_3O_8" is used here, and in a similar connection in other pages, as a basis of calculation, because it is the basis now universally used in commerce when ores are purchased or sold. Strictly speaking, the symbol should be "UO_3."

A. LOWER PART OF SETTLING TANKS, WITH EARTHENWARE FILTERS BELOW

B. UPPER PART OF SETTING TANKS, SHOWING FLOATING SIPHON

dium and too much uranium. Not only is the amount of sodium carbonate in excess important but also the time of boiling after the acid has been completely added. In actual practice, the boiling is continued for three hours after the last of the acid has been run into the tank. There are, therefore, two important factors that control the amount of uranium and vanadium left in the precipitate, namely, the excess of sodium carbonate and the period during which boiling is continued, the latter being just as important as the former.

FILTERING THROUGH PRESSES.

The solution is then filtered through filter presses—in the first plant through press *33a* (Pl. IV); in the second plant through press *16* (Pls. VI and VII), the filtrates running into tanks *6* (Pls. III, VI, and VII), in which the uranium is afterwards precipitated. The precipitates are washed in the filter press for about 20 minutes, the washing being added to the original solution. It is, of course, advisable to wash the precipitates until they are as free as possible from adhering liquor so as to decrease the loss on uranium and vanadium, but the amount of washing is controlled by the capacity of the tanks into which the filtrate is run.

The uranium and vanadium content of the iron precipitate obtained has varied considerably, the average uranium content running about 0.7 per cent U_3O_8, and the average vanadium content about 2 per cent V_2O_5. Some of the precipitates have gone as low as 0.5 per cent U_3O_8 and 1 per cent V_2O_5, whilst once or twice, when there was a temporary lack of steam and the boiling was not continued sufficiently long, the uranium content of the iron precipitate was more than 2 per cent U_3O_8.

RECOVERY OF BARIUM SULPHATE PRECIPITATE.

After the acid liquor has been added to the sodium carbonate tank, as described above, and before the filter-pressing of the iron precipitate, the radium-barium sulphate precipitate in the conical settling tanks is run through earthenware stopcocks at the bottom of each settling tank onto the suction filters. The solution is filtered under suction, and, by means of a rubber hose attached to the spout of the filter, the filtrate is added to the sodium carbonate tank. The sulphate is washed with distilled water, and with a weak solution of sodium hydroxide, in order to neutralize any adhering traces of acid. It is then carefully scraped from the filter paper placed over the asbestos filter cloth and is put into an iron pan 19 by 20 by 3 inches. The sulphate is dried in a hot-air oven heated by steam coils, after which it is transferred to the sulphate building for further treatment. If the solution at any time runs through cloudy, owing to the

presence of barium sulphate in the filtrate, it is run back into tank *1* (Pls. III, IV, VI, and VII) by means of a long rubber hose, and the sulphate that has gone through the filter is thus recovered with the next radium-barium sulphate precipitate. It is found preferable to do this rather than to try to refilter the solution at once. Such mechanical losses in plant operation should be carefully watched, as they are more likely to occur than chemical losses.

RE-SOLUTION OF IRON PRECIPITATE.

When the first plant was started it was thought that not only would the recovery of the uranium and vanadium content of the iron precipitate be justified, but also that the radium not recovered in the first radium-barium sulphate precipitate might be sufficiently large to justify its recovery also. Therefore, the original plan of procedure involved the re-solution of the iron precipitate in hydrochloric acid and the precipitation of a second radium-barium sulphate precipitate, followed by a second precipitation with sodium carbonate.

Accordingly, into tank *2* in the first plant was poured a sufficient amount of commercial hydrochloric acid to just dissolve the iron precipitate. This was diluted with an equal volume of water. The amount of acid required varied with the ore treated. Some ores carry more iron than others, and, therefore, the exact quantity of hydrochloric acid required in all cases can not be stated. In addition, if the iron precipitate carried a little more vanadium than usual, this affected the quantity of acid required. Generally speaking, however, 950 pounds of 28 per cent hydrochloric acid was required for each ton of ore treated. The acid was used cold, the iron precipitate being added slowly, one shovelful at a time, the liquor being thoroughly agitated by a paddle during the addition. If the iron precipitate is added too rapidly, the solution heats, and toward the end of the reaction there is a tendency for the vanadium to separate. Under these conditions it is difficult to get the whole mass in solution, so that when the radium-barium sulphate is precipitated, an unsatisfactory, dirty sulphate is obtained.

When all the material is in solution the tank is almost filled with water and 3 pounds of barium chloride added. This is followed by the addition of 30 to 40 pounds of 66° B. sulphuric acid. The dilution and the amount of sulphuric acid added is affected, to some extent, by the amount of calcium in the ore, as calcium sulphate is likely to be precipitated if the solution is not sufficiently dilute.

The radium-barium sulphate thus obtained was elevated to one of the settling tanks and allowed to settle until the next day. The clear liquid obtained was siphoned off by means of a floating siphon, in the ordinary way, and run into an excess of hot sodium carbonate,

the iron-calcium precipitate obtained being filter-pressed, and the uranium and vanadium in the filtrate recovered.

It was found that the cost of recovering this uranium and vanadium was greater than the value of the products obtained, and after a few trials, the second precipitation of uranium and vanadium was abandoned, and the tanks that were designed for use in this part of the process, namely, tanks *2*, *4*, *5*, and *7* (Pls. III, IV, *B*, and V) were afterwards used for other purposes. However, the second precipitation of the radium was continued for a longer period of time, "second" sulphates being obtained on the first eight cars of ore treated. Later, it was found that the recovery on one car (P–7) was not satisfactory, and that some of the radium was going through in the filtrate from the first radium-barium sulphate precipitate. The re-solution of the iron precipitate and the formation of second sulphates was again started at this time and continued with the ore from three cars (P–7, P–8, and P–9). As the cause of the losses was then ascertained to be of a mechanical nature, and eliminated, the formation of the second sulphates was again discontinued, as the amount of radium obtained in the sulphates did not justify the expense.

PRECIPITATION OF URANIUM.

The filtrate from the iron precipitate carries the uranium in solution as a double sodium uranyl carbonate and the vanadium as sodium vanadate. The solution is partly neutralized, acid being added until a yellow precipitate begins to form. This precipitate is supposed to be uranyl carbonate, but in reality it contains more sodium uranate than uranyl carbonate, as on ignition it gives a very small amount of oxide. If too much acid is added at this stage the amount of vanadium that appears with the sodium uranate is considerably increased. When the precipitate begins to foam the addition of acid is stopped and sodium hydroxide is added to the hot liquor until the uranium is completely precipitated as sodium uranate. The completeness of the precipitation is easily tested by filtering off some of the precipitate on a small funnel and adding more sodium hydroxide to the filtrate in the test tube or beaker.

During the operation of the first plant, before the second had been built and before a nitric-acid plant had been erected, sulphuric acid was used to partly neutralize the sodium carbonate. The sulphuric acid was run into the iron tank (*13*, fig. 4) and elevated by compressed air into the lead-lined tank (*41*, fig. 4), where it was diluted and run through lead pipes to the uranium-precipitating tank (*6*, Pl. III) and the vanadium tanks (*7*, Pl. III, and *39*, fig. 4 and Pl. V). Therefore the filtrate from the vanadium precipitate afterwards obtained contained not only sodium nitrate but also large quantities of sodium

sulphate, and it was necessary to separate the sodium sulphate by means of fractional crystallization, as described subsequently. After the nitric-acid plant had been erected it was found much more satisfactory and, in the end, cheaper to neutralize at this point with nitric acid instead of with sulphuric acid, so that the grade of sodium nitrate obtained could be greatly increased and fractional crystallization could be eliminated.

After the addition of the sodium hydroxide and the precipitation of the uranium the solution is boiled for one hour in order to promote complete precipitation and is then filtered through press *17* (Pl. VI). This press takes care of the uranium in both plants, although origin-

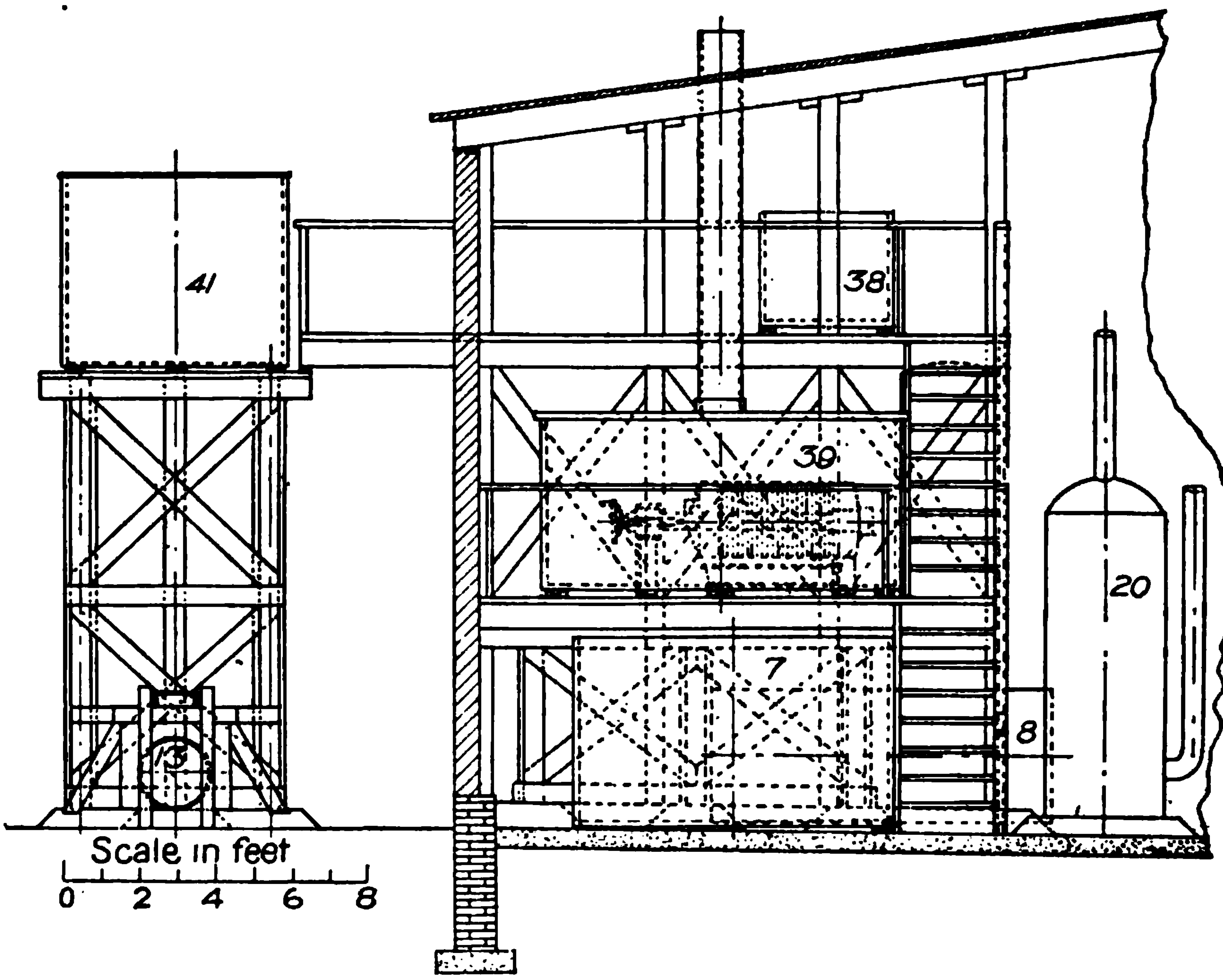


FIGURE 4.—Section of first, or "old," plant. *7*, vanadium tank; *8*, distilled-water tank; *13*, sulphuric acid elevating tank; *20*, steam baffle; *38*, ferrous sulphate tank; *39*, vanadium press; *41*, sulphuric acid tank.

ally the uranium precipitate in tank *6* (Pl. III) in the first plant was filtered through press *33c* (Pl. IV, *A*). The sodium uranate cake is washed for about 15 minutes and then dried in pans in hot-air ovens (*22*, Pls. V and VII). This method of drying has been found to be the best under the circumstances, not only for the radium-barium sulphate precipitates, but also for the uranium and the vanadium.

Trial was made of another method, embracing the use of large iron pots heated by direct heat from a small fire placed beneath, but the results were not satisfactory; both the uranium and the vanadium precipitate caked readily and dried slowly, notwithstanding considerable hand stirring. The method, therefore, was abandoned.

As the sodium uranate requires re-treatment, owing to the fact that it carries vanadium, it is not necessary to wash the cake as completely as might otherwise be required. Most of the sodium uranate carries 7 to 9 per cent V_2O_5. It has been found practically impossible to obtain a precipitate that does not carry considerable quantities of vanadium, most of which appears probably as uranium vanadate.

PRECIPITATION OF VANADIUM.

The filtrate from the sodium uranate precipitate contains the vanadium as sodium vanadate. The solution is brought to the boiling point and just neutralized with nitric acid, the boiling being continued long enough to eliminate the carbon dioxide. A workman, after a little practice, is able to neutralize the solution and do the rest of the work satisfactorily. A solution of ferrous sulphate, made by dissolving ordinary commercial ferrous sulphate in cold water, is then run from small storage tanks (*38*, fig. 4 and Pls. V and VII) into the hot solution, agitation being accomplished by means of compressed air. The amount of ferrous sulphate added depends to some extent upon conditions, such as acidity, irrespective of the amount of vanadium present. As a rule, about 75 pounds of ferrous sulphate is required per ton of ore treated. The heating of the solution is stopped before the addition of the ferrous sulphate, for if heating is continued longer a complete precipitation of the vanadium is not obtained.

It is advisable to have the solution just neutral after the addition of the sulphate, and if it is neutral before this addition it will of course be slightly acid afterwards. It is difficult so to gauge the amount of acid added to the vanadium solution that the latter will be exactly neutral after the addition of the ferrous sulphate. In practice it is found convenient to make the solution exactly neutral before the addition of the ferrous sulphate and then to add a few pounds of sodium hydroxide to neutralize the solution once more after the addition of the ferrous sulphate. In this way practically all of the vanadium is precipitated as iron vanadate, only a mere trace going through in the filtrate. The precipitate is probably a mixture of the different vanadates of iron and has a greenish-gray color; it usually contains 32 to 33 er cent of V_2O_5.

It is not difficult to control the grade of the precipitate obtained. If the solution is slightly alkaline before the addition of the ferrous sulphate, a product carrying as low as 25 per cent V_2O_5 may be obtained. This is brown, with practically no green tinge. If the solution is faintly acid after the addition of the ferrous sulphate, a product may be obtained that is somewhat yellow and under favorable conditions may carry as much as 42 per cent V_2O_5. In other

words, in a slightly acid solution, with continued boiling, a high-grade product is precipitated, but all of the vanadium is not recovered, some of it going into the filtrate. This contaminates the sodium nitrate and undoubtedly causes losses on evaporating the nitrate solution owing to decomposition.

If carnotite is treated in a beaker with nitric acid, filtered, the filtrate poured into an excess of hot sodium carbonate solution, the uranium precipitated by sodium hydroxide, and the vanadium precipitated by ferrous sulphate the product universally obtained is a dark-brown precipitate carrying less than 25 per cent V_2O_5. On a large scale oxidation is much more complete, and a much higher grade product can be obtained than in the laboratory. Seemingly this result is due not only to the long time during which the liquids are boiled, but also to the fact that air is being continually passed through them.

Before the erection of the nitric-acid plant the neutralization of the vanadium solution was accomplished by means of sulphuric acid instead of nitric acid. The change was made for reasons already indicated under the discussion of uranium precipitation.

NITRATE RECOVERY.

Before the second plant had been built, the evaporator and the crystallizing pans for the recovery of the sodium nitrate were placed in the angle formed by the first plant and the boiler house, which is now used as a furnace room. After the second plant had been added, the crystallizing pans were removed from their original position to that shown in Plate VI (*44*). An extra evaporator was also installed to take care of the increased capacity. The present equipment handles all of the nitrate solution from both plants.

As already stated, while the first plant was being operated by itself, the excess sodium carbonate in the uranium and vanadium tanks was neutralized by means of sulphuric acid, consequently the filtrate from the vanadium precipitate consisted of a neutral solution of sodium nitrate and sodium sulphate. It was necessary, therefore, in order to recover a sufficiently high grade of nitrate, to separate a considerable amount of the sulphate from the nitrate by fractional crystallization.

The filtrate from the vanadium precipitate was stored in tanks *11* and *12* (Pls. III and V) before the liquor was run into the evaporating pan. One of these tanks was fitted with a grid made of Byers pipe. Waste steam was run through the grid so that partial evaporation took place in the tank itself before the liquor was run into the evaporating pan, in which it was finally concentrated to a specific gravity of 1.23 to 1.27. As concentration was effected in the pan, more liquor

was pumped from the storage tank. When a sufficient amount of the concentrated liquor had accumulated, it was run into a crystallizing pan (Pl. XI, *A*). The product obtained from this first crystallization, consisting mostly of sodium sulphate, varied somewhat in composition, the variation depending largely on the temperature of the air during crystallization. Usually, however, the wet salt carried between 3 and 10 per cent of sodium nitrate, the water content varying from 42 to 50 per cent, and was classed as "high-grade sulphate." As it could not be sold or even given away in Denver, most of it was dumped.

The mother liquor from the crystals was pumped back into the evaporator and concentrated to a gravity of 1.35 to 1.38, and allowed to crystallize. The product obtained from this crystallization was classed as "low-grade nitrate," and as with the first crystallization, the composition varied somewhat, according to the temperature of the air during the crystallization. Most of the wet product carried 30 to 35 per cent sodium nitrate and 20 to 30 per cent water. The liquor from the low-grade nitrate was further concentrated to a gravity of 1.40 to 1.44 and again allowed to crystallize. The material obtained from this crystallization was classed as "high-grade nitrate," and the wet salt contained 65 to 80 per cent sodium nitrate and between 6 and 10 per cent water. The mother liquor from these crystals was added to the next batch of liquor from the low-grade nitrate.

USE OF LOW-GRADE NITRATE UNSATISFACTORY.

In the making of nitric acid from this recovered nitrate it was necessary to work in the low-grade nitrate in small quantities with the high-grade nitrate or fresh purchased nitrate. Even under these conditions the amount of moisture and sodium sulphate present in the low-grade nitrate made its use unsatisfactory. When the losses were taken into consideration, as well as the cost of the additional sulphuric acid required because of the presence of the large amount of sodium sulphate in the nitrate, it was found not only more satisfactory, but actually cheaper, to use nitric instead of sulphuric acid in the neutralization of the sodium carbonate in the uranium and vanadium tanks. Under these conditions it is not necessary to crystallize out any sodium sulphate, as the amount present is small, coming from the ferrous sulphate and also the sulphuric acid used in the precipitation of the radium barium sulphate. The nitrate obtained under these conditions usually runs from 80 to 85 per cent $NaNO_3$.

STORAGE OF NITRATE SOLUTION.

According to the present arrangement, tanks *11* and *12* in the new plant (Pl. VI) are used as storage tanks for the nitrate solution from both plants, as the vanadium precipitate from both plants is filtered through filter press *18* (Pls. VI and VII) in the new plant. Partial evaporation takes place in both tanks, waste steam being used. The liquor is then pumped from the tanks into the evaporators (*43*, Pl. VI), where evaporation is carried on until a gravity of 1.4 is obtained. At this gravity crystals will usually separate out to some extent in the hot liquor, and on running the liquor into the crystallizing pans crystals are frequently found on the bottom of the evaporator.

The eight crystallizing pans are divided into two groups of four, each group taking care of one day's run from both plants, so that crystallization takes place over a period of 48 hours. The mother liquor from the first crystals is pumped through a Worthington pump back into the evaporator, this pump being the same as is used to pump the liquor from the reservoir tanks into the evaporators. On the second evaporation the gravity is run to between 1.44 and 1.46, the mother liquor from these crystals being added to the next batch of fresh material.

The sodium nitrate obtained after the removal of the mother liquor is shoveled out onto the draining boards which are built over the crystallizing pans. After thorough drainage the nitrate is removed to the nitrate storage house, situated between the crystallizing pans and the nitric-acid plant.

METHOD OF USING CRYSTALLIZING PANS.

The actual procedure in using the different crystallizing pans is as follows:

The mother liquor from pans *1*, *2*, *3*, and *4* (Pl. III), or in reverse order, is pumped into one of the evaporators as needed. After all of the mother liquor from the four pans has been introduced into the evaporator, the crystals from the previous crystallization are shoveled from the pans onto the draining boards (Pl. XI, *A*). The liquor in the evaporator is then run into pan *1* for the second crystallization. Sometimes the volume of this liquor is large enough to make the use of pan *2* also necessary. The nitrate solution, which has been more or less concentrated in the storage tanks (*11*, *12*, Pls. VI and VII), meanwhile has been further concentrated in the other evaporator. When a gravity of 1.4 is reached, the liquor is run into pans *2*, *3*, and *4* for the first crystallization. The next day the same process is repeated with pans *5*, *6*, *7*, and *8*, the mother liquor from the second

A. EVAPORATION AND CRYSTALLIZING PANS FOR RECOVERY OF SODIUM NITRATE

B. OIL FURNACES FOR REDUCTION OF SULPHATES AND REFINING OF SODIUM URANATE

crystallization in each instance going back into the fresh material from the plant.

Since both plants have been running full capacity, the average weight of sodium nitrate recovered per day has been close to 4,000 pounds on the 100 per cent basis. This, of course, has varied to some extent, as the recovery has varied also. The average analysis of the nitrate during this period has been as follows:

Average composition of nitrate solution.

Constituent.	Proportion in the solution.		
	Maximum.	Minimum.	Average.
	Per cent.	*Per cent.*	*Per cent.*
Sodium nitrate	85.00	83.38	84.33
Sodium sulphate	4.80	3.99	4.39
Sodium chloride	3.96	2.32	3.16
Water	8.25	7.08	7.94

NITRIC ACID MANUFACTURE.

A plan of the nitric acid plant is shown in Plate XII. The sodium nitrate is elevated by means of the elevator *c*, and is bedded and sampled at *b*. As the composition of the nitrate varies to some extent, every day, it is necessary to make a daily analysis of the nitrate used. The charge is wheeled in wheelbarrows from *b* to the stills *d*. The bleacher and the Hart condensers are shown at *e*, the towers at *f*, and the storage pots at *g*. As the manufacture of nitric acid is more or less standardized, it is not necessary to describe it in detail in this report. The plant has produced an average of 5,250 pounds of acid per day over a period of several months, the whole of the acid averaging 63 per cent in strength.

URANIUM REFINING.

As already stated under the description of the precipitation of uranium, most of the sodium uranate that is obtained contains 7 to 9 per cent of V_2O_5 on the dry basis. It has been impossible to precipitate sodium uranate in alkaline solution containing both uranium and vanadium and not precipitate at the same time a considerable amount of vanadium with the uranium. As uranium should be as free from vanadium as possible, the economical refining of the uranium is important.

METHODS FIRST USED.

The first attempts to remove the vanadium from the sodium uranate were along the line of reprecipitation. The sodium uranate was redissolved in hot dilute sulphuric acid and sodium hydroxide

added in sufficient excess to reprecipitate the uranium. It was found that one precipitation carried on in this manner reduced the amount of V_2O_5 in the sodium uranate by about 50 per cent. A second reprecipitation reduced the amount of V_2O_5 that was left in the product again to about half, so three, and sometimes even four reprecipitations would be necessary to reduce the V_2O_5 content below 1 per cent. Such a procedure, of course, would not be commercially feasible. Consequently, it was necessary to find some other method that would be more economical.

If sodium uranate containing vanadium is heated with somewhat concentrated nitric acid, evaporated almost to dryness and then treated with water, the larger part of the vanadium remains undissolved, but the vanadium carries considerable uranium, and the uranium still retains some of the vanadium. If more dilute acid is used and boiling continued for only a few minutes, practically all of the vanadium can be precipitated, but the precipitate still retains a considerable amount of uranium. Somewhat the same results can be obtained with hydrochloric or sulphuric acid, but with either, the precipitate of vanadium carries down considerable quantities of uranium.

RESULTS OBTAINED BY VARIOUS INVESTIGATORS.

Smith and Gibbs [a] have shown that vanadic acid can be removed from heated sodium uranate by means of hydrogen chloride, the vanadium volatilizing. Hillebrand [b] has shown that vanadium also can be partly removed in the same manner from ores. In the case of the sodium uranate, the product left behind is probably a mixture of sodium chloride, uranyl chloride, and sodium uranate. Barker [c] obtained similar results. He also removed vanadium from sodium uranate by mixing the uranate with twice its weight of ammonium chloride and enough water to make a thick paste. On heating, the vanadium may be practically completely volatilized, the amount left in the residue being reduced to as low as 0.5 per cent. The ammonium chloride at the same time converts the uranium present to ammonium uranate, which yields uranium oxide. If the temperature is too high, the amount of oxide obtained is reduced materially, as some of the oxide is converted back to sodium uranate. The best conditions are obtained when the temperature is not raised higher than is necessary to volatilize the vanadium and the ammonium chloride.

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[a] Jour. Am. Chem. Soc., vol. 16, 1894, p. 578.

[b] Hillebrand, W. F., and Ransome, F. L., Carnotite and associate minerals: Bull. 262, U. S. Geol. Survey, 1905, p. 9.

[c] Barker, H. H., Unpublished thesis.

Any of these methods might be used for the removal of the vanadium, but on a commercial scale practically all of them involve serious difficulties. The use of nitric, hydrochloric, or sulphuric acid would give excellent results, provided the precipitation between the vanadium and uranium was sharp and complete, but if all the vanadium is to be removed such a separation is practically impossible. Too much uranium goes with the vanadium, and if the loss of uranium is reduced, a considerable amount of vanadium is not separated from the uranium.

FUSION METHOD.

The authors have found, however, that the vanadium can be separated from sodium uranate cheaply and efficiently by fusion methods. If sodium uranate is heated with two to two and one-half times its weight of sodium sulphate until the whole mass is fused, and the product is afterwards broken up and leached, practically all of the vanadium goes into the solution and the uranium still remains as insoluble sodium uranate. In this manner the vanadium content of sodium uranate can be reduced from 8 or 10 per cent to less than 0.5 per cent by one treatment, practically all of the vanadium being removed if the product is sufficiently washed.

The main trouble with this method is in finding a satisfactory container for the material during fusion. Steel is slightly attacked by fused sodium sulphate, and cast iron, although standing up much better under the action of the molten material, has too low a melting point to make its use advisable for straight sodium sulphate. Although the sulphate has a melting point much lower than that of cast iron, when considerable quantities of sulphate are used it is difficult to get the center part of the mass melted before the exterior reaches the melting temperature of cast iron. Vitrified brick stands up well, and therefore a small reverberatory furnace could be used. The authors have found it more convenient, however, to add a small quantity of sodium nitrate to the sodium sulphate, in order to lower the melting point of the sulphate. If 20 to 25 per cent of sodium nitrate is added to the sodium sulphate, a melt can be readily obtained in a cast-iron pot.

USE OF OIL FURNACES.

Oil furnaces with Case burners (Pl. XI, *B*) have been used at the plant of the National Radium Institute, the pots holding 150 pounds of the mixture of sodium nitrate and sodium sulphate. To this melt is slowly added from a hopper 40 to 50 pounds of sodium uranate, with frequent stirring by means of an iron rod flattened at the end and inserted in a wooden handle. Care must be taken that effervescence does not occur too rapidly, as otherwise the pot may boil over.

After the sodium uranate has been completely added, heating must be continued for about one hour longer, in order that all of the material may be acted upon by the melt.

It is run out through a spout at the bottom of the pot into a suitable iron container, broken up, run through a ball mill, either dry or, preferably, with water, and leached with boiling water in a suitable tank. The leaching is done in tank *4* (Pls. III and IV, *B*) in the first plant, and the product pressed through filter press *33b* (Pl. IV), the filtrate running into tank *7* (Pl. III), in which the vanadium is precipitated. The sodium uranate thus obtained, provided the fusing has been carried out properly, is practically free from vanadium.

FUSION WITH SODIUM SALTS.

This procedure fitted in excellently with the main process, as sodium sulphate containing sodium nitrate was one of the by-products in the crystallizing of the sodium nitrate, and therefore purchase of the material was not necessary. Other sodium salts besides sodium sulphate, or a mixture with sodium nitrate, have a similar effect. Fused sodium hydroxide, sodium carbonate, or a mixture of sodium hydroxide and sodium carbonate, removes vanadium from sodium uranate, the sodium uranate being insoluble in a leaching process, but neither the hydroxide nor the carbonate gives as good a result as sodium sulphate. Ordinary salt, however, is just as efficient as sodium sulphate. In addition it has the advantage of not attacking cast steel, so that steel pots may be used and the salt readily melted therein. For this reason salt was substituted for the mixture of sodium sulphate and sodium nitrate at the Radium Institute.

"LIGHT" AND "DARK" SODIUM URANATE.

The sodium uranate that is obtained by this fusion method is off color. In order to obtain the light-yellow material, re-treating would be necessary. If the uranium is present as double sodium uranyl carbonate, the color of the product can be controlled without much difficulty. If acid is added to the solution until neutralization is almost complete, the precipitate that comes down is light yellow, sometimes almost white. If before neutralization is actually completed, sodium hydroxide is added until all of the uranium is precipitated, the color of the precipitate will be light yellow. The same result can be accomplished by adding acid until the solution is slightly acid, and then adding sodium hydroxide until precipitation is just complete. In order to get the dark variety of sodium uranate by direct precipitation, all that is required is to add sodium hydroxide to the solution of sodium uranyl carbonate without previous partial or complete neutralization. It is also advisable not to wash too thoroughly the sodium uranate obtained in this manner. Unless

there is no objection to the presence of vanadium the precipitation of part of the vanadium along with the uranium, of course prevents the direct handling of the material in the manner described.

CONVERSION OF SODIUM URANATE INTO URANIUM OXIDE.

The conversion of sodium uranate into uranium oxide is also of interest. The first work of this kind done by the authors involved precipitation with ammonium hydroxide in order to obtain ammonium uranate, so that on ignition of the uranate U_3O_8 could be obtained. Boiling the sodium uranate with a strong solution of ammonium chloride was also tried. Some laboratory experiments were carried out on these methods before they were used in the plant, with the following results:

A 5-gram sample of dry sodium uranate was dissolved in 3 c. c. of concentrated sulphuric acid diluted with 25 c. c. of water, and the solution was poured into 25 c. c. of ammonia, the total volume being made up to 100 c. c. Excess of ammonia was expelled by boiling until rather violent bumping occurred. The yellow precipitate was filtered off and washed twice on the filter and ignited. Another sample of the same weight was boiled with 50 grams of ammonium chloride in 50 c. c. of water. The liquid was boiled for half an hour, a drop of ammonia being added from time to time to maintain an alkaline solution. The precipitate was treated as before. The weight of the residue obtained from the first experiment was 3.99 grams, and from the second 3.85 grams.

Each residue was digested with 1 c. c. of sulphuric acid in 25 c. c. of water, the weight of the residue obtained from this treatment being 2.47 grams for the ammonia-treated material and 2.71 for the other. The difference does not represent solely the amount of sodium uranate with the uranium oxide, as under the conditions at least 15 per cent of U_3O_8 is dissolved by the acid. For the final product, U_3O_8 is what is desired, and the experiments showed that by the use of a large excess of ammonium chloride a yield of 54 per cent of U_3O_8 is obtained, and of 50 per cent by digestion in ammonium sulphate in excess.

Further experiments showed that 2 parts by weight of ammonium chloride to 1 part of uranate gave practically as good a yield as when 5 parts of the chloride were used, and the yield was not very much reduced when the amount of chloride was still further decreased so that the ratio was 1.5 to 1.

EFFECT OF REDUCTION.

In the course of the experiments it was observed that in dissolving the sodium uranate in concentrated sulphuric acid and in passing from sodium uranate to the oxide a partial reduction to uranous

salts took place. It was therefore thought possible that the efficiency of the conversion might be increased by reducing the sodium uranate before its precipitation as the ammonium salt. The reduction was accomplished by adding to the sulphuric acid a solution of sulphurous acid and boiling to expel the excess of sulphur dioxide. The uranium was then precipitated by ammonium hydroxide in the ordinary way. The efficiency showed a slight increase, but the difference was within the limits of experimental error.

The results of the work clearly indicated that it is difficult to completely convert sodium uranate into ammonium uranate by one treatment with ammonium chloride or ammonium sulphate, the conversion into ammonium uranate being usually somewhere around 60 per cent. In other words, a mixture of ammonium uranate and sodium uranate is obtained. On ignition the ammonium uranate is, of course converted into oxide, which remains contaminated with the unconverted sodium uranate. The latter can be leached out by dilute sulphuric acid, the preferable strength being 4 or 5 per cent. In this manner a fairly pure oxide can be obtained, which, however, usually contains traces of sodium uranate, iron, silica, and aluminum.

PLANT PROCEDURE TRIED.

The actual procedure which was finally used in experimental work in the plant was as follows:

One hundred pounds of the refined sodium uranate was dissolved in 100 pounds of 66° B. sulphuric acid diluted with 200 pounds of water. The solution took place in an earthenware pot (*34*, Pl. IV). The sodium uranate dissolved, but the majority of the iron oxide, as well as some other impurities, remained insoluble. Most of the iron was derived from the impurities in the sodium sulphate and sodium nitrate used in the fusion method for eliminating the vanadium, as already described. The iron oxide could be removed by filtration through an earthenware filter (*35*, Pl. IV) or could be run direct into the precipitation tank (*5*, Pls. III and IV, *B*), depending upon whether it was desired to eliminate the iron, and thus obtain a higher-grade product. If the solution was not filtered, the oxide finally obtained was about 87 per cent U_3O_8. If the solution was filtered at this stage, the grade of the final product could be greatly increased, reaching as high as 95 per cent.

The solution was then run into tank *5* (Pls. III, and IV, *B*), which contained about 5,000 pounds of water, and an amount of ammonia slightly in excess of that required to neutralize the acid. After 300 pounds of sodium uranate had been treated in the manner described and run into tank *5*, the liquid in this tank was boiled for about two hours. It was then filtered through press *33c* (Pl. IV), and the

filtrate discarded. If this method were used continuously on a commercial scale, it would, of course, pay to treat the solution and recover the ammonia so that it could be used again.

The washed precipitate was then removed from the press, dried, and ignited in an iron pot in one of the oil furnaces in the furnace room. In this manner, the ammonium uranate was converted into oxide. As the product obtained, as already described, consisted of a mixture of uranium oxide and sodium uranate, it was treated with a 5 per cent solution of sulphuric acid, the sodium uranate dissolving along with a small amount of oxide. The solution was run into tank *5* and retreated.

SMALL-SCALE TESTS OF OTHER PROCESSES.

Although this process worked fairly well, the costs were altogether too large to justify its permanent use. Other methods were therefore sought whereby a high-grade oxide could be obtained and the vanadium eliminated at the same time, with a decrease in the cost of conversion. In other words, a method was required by which uranium oxide could be cheaply and efficiently produced from the sodium uranate without any preliminary treatment to get rid of the 7 or 8 per cent of V_2O_5 always present. Following a suggestion made to one of the authors it was found that when sodium uranate was ignited in a platinum crucible to a temperature approaching 1,500° C., a certain part of the uranate was converted into oxide. During the heating, fumes of sodium oxide were evolved, and when the product obtained was leached with water, the water was found to be strongly alkaline and contained practically all of the vanadium that was in the sodium uranate. Some experiments were tried on a semicommercial scale, a small oil furnace containing a shelf of fire-clay on which the sodium uranate was placed being used. The flame was applied not only underneath the shelf but also over it, giving a more or less reverberatory effect. A temperature approaching 1,500° C. was obtained, and a considerable amount of sodium uranate was converted into oxide, but the conversion was by no means complete. In addition, the fire clay was attacked by the semiviscous mass. The conclusion was then reached that the only possibility of success along this line was to use an electric furnace.

TESTS OF VARIOUS CRUCIBLE MATERIALS.

It was also important to find some material that would not be seriously affected by the action of the melted sodium uranate. Among others, the following were tried:

An Acheson graphite crucible brasqued with magnesite; a similar crucible brasqued with alundum; a similar crucible brasqued with 50

parts of magnesite and 50 parts of alundum; a crucible made of silicon-carbide tubing bonded with carbon, with a graphite bottom luted in place with silicon carbide and water glass, and the inside of the crucible coated with silicon carbide and water glass; a crucible similar to the last except that quartz tubing was put in with the charge.

Alundum proved to be rather unsatisfactory, as it was quickly attacked. The magnesite lining separated fairly well from the charge, but the charge was to some extent contaminated. The charge also stuck badly to the lining of silicon carbide and water glass, and the result was that practically none of these materials appeared to be satisfactory. The experiments were carried on in a resistance furnace, with a granular carbon resistor in which the crucibles, or tubes, were embedded.

In another experiment 200 grams of sodium uranate was placed in a small Acheson graphite crucible with a direct arc from a carbon rod, constituting a small Héroult type furnace. In this, the whole charge fused, and copious clouds of alkaline fumes were given off. The product was black, and the vanadium could be readily leached out when the ground material was boiled with water. Similarly an iron pot cooled with water externally was used as one electrode and container, the other electrode being made of Acheson graphite. With this equipment several pounds of a fairly satisfactory black oxide was obtained. The results of this experiment were so encouraging that work was continued along this line in the plant.

CONSTRUCTION OF EXPERIMENTAL ELECTRIC FURNACE.

The mechanical part of the experimental furnace consisted of a heavy 4-inch L-beam 10 feet long placed horizontally 8 feet from the floor, one end being embedded in the wall and the other bolted to a 3 by 4 inch wooden support embedded at its lower end in the cement floor. At the middle of this horizontal beam, two plates, or hangers, 2 feet by 4 inches by ½ inch were suspended. In the lower ends of these a 1-inch shaft carrying the sprocket wheels was placed horizontally. Over one sprocket wheel passed a chain having one end connecting with the electrodes and the other end off to one side passing over a wheel and terminating in a counterbalance approximately the same weight as the electrodes. Over the other sprocket wheel on the shaft in the central hanger passed a continuous chain about 10 feet long, running also on a sprocket wheel placed on a 1-inch shaft attached to the upright wooden support of the main horizontal L-beam. Attached to this 1-inch shaft was a 22-inch hand wheel, the turning of which raised and lowered the electrodes.

On the floor directly beneath the electrodes was placed a cement basin about 2½ feet square and 10 inches high, sloping inward toward

the bottom and drained by a 2-inch pipe. For melting the sodium uranate an iron pot standing on legs was placed in the basin. The pot was cooled by water from a ring placed about 2 inches from the top and having $\frac{1}{16}$-inch holes every half inch. About 3 inches of water was allowed to back up into the basin to cool the bottom of the pot. Around the inside of the basin, stones and bricks were packed loosely about halfway up the pot to minimize danger from an explosion, if any of the molten material should be thrown out of the pot. These loose stones and bricks were kept away from the pot by a sheet-iron sheath around the pot, being enough larger to allow free play for the cooling water to run down the sides of the pot.

VOLTAGES USED.

The voltage first used was 170 volts, but after several trials the arc was found to be too heavy and to sputter too much. Other voltages ranging from 80 to 130 were tried, 130 volts proving the most satisfactory, although a 110-volt pressure was used to some extent. Leads of 600,000 circular mils about 20 feet long extended from the transformers, which were not sufficiently flexible to be connected to the electrodes, so that they were spliced to 5-foot lengths of flexible leads, $\frac{1}{2}$ inch by 2 inches in cross section, made of soft-copper strips wrapped with tape.

In order to start the arc, it was found that the best method was to use a piece of uranium oxide from a previous charge, which was heated in an oil blast to bright redness and placed beneath and in contact with the electrodes. Such material is sufficiently conducting to melt and form the desired pool.

MELTING POTS USED.

The preliminary work was done in cast-iron pots, which, however, did not stand up under the treatment, and usually, after one or two runs, were melted through at some point. In order to make the pots last longer, a coating of fire clay on the inside was tried, magnesite being afterward substituted. The molten material, however, attacked, and to some extent dissolved both the fire clay and the magnesite; consequently, their use was abandoned. Finally, the plan already described was adopted, namely, the use of a heavy pot made of $\frac{3}{8}$-inch boiler plate 14 inches in diameter and 10 inches on the side, with a round bottom, dipped 2 inches, and standing on legs, so that the lowest point of the bottom stood half an inch from a level floor. The pot was riveted and worked well when it was cooled by a stream of water from a circular pipe that surrounded it, as already described. The pot would hold about 160 to 175 pounds of product, but was unwieldy to remove and dump, so ladling was resorted to at inter-

vals. This resulted in a yellow coating appearing on the material, which was probably re-formed sodium uranate, increasing the percentage of this impurity in the product. In order to eliminate this trouble as much as possible, a second pot was made, and after the charge had cooled the pot with contents was removed and the second one substituted.

PRODUCT OBTAINED.

The product finally obtained consisted of UO_2, containing small traces of undecomposed sodium uranate, especially on the top and sides of the charge, and also some Na_2O that had not been vaporized. After the product had been put through a ball mill and leached with water, the filtrate contained most of the vanadium that had been in the original sodium uranate, and this was recovered in the usual way. The leaching also washed out the soda which was present.

In order to eliminate the trace of sodium uranate, it was necessary to wash with dilute sulphuric acid, sodium uranate being soluble in a 4 or 5 per cent acid, whilst the oxide is not. Uranium oxide (UO_2) is much less soluble not only in sulphuric acid but also in hydrochloric acid than U_3O_8, the amount of oxide going into solution with this treatment being small.

EXPLOSIONS IN ELECTRIC FURNACE.

Explosions in the melt, which quickly developed in connection with the use of the electric furnace, finally forced its abandonment. The explosions were of two distinct types—one like a sharp pistol crack, the other of full, round, cannon-boom character. The first frequently occurred alone, but the latter was always preceded by a sharp explosion. The sharp explosions evidently came from the melt, as they always threw molten material from the pot. At times the furnace could be operated for a considerable period smoothly and satisfactorily, when suddenly loud "cracks" would be heard, followed by molten material being thrown in all directions. Sometimes these cracking explosions would occur every 15 or 20 minutes. At first the authors were rather inclined to believe that the explosions were due to nitrate left in the sodium uranate, but later experiments with material washed perfectly free from nitrate gave the same results. Two or three of the explosions were violent enough to tear the hood from the bolts, break the "transite" asbestos boards, and throw the bricks out of the cement basin.

The possibility of the "booming" explosions being due to an explosion of carbon monoxide formed from the electrodes was investigated. Samples of gas were taken from just over the pot at several intervals and analyzed. In each sample mere traces of carbon monoxide or hydrogen were present; consequently, this explana-

tion had to be abandoned. In addition, the draft through the hood was so excellent that there was no possibility of an accumulation of a sufficient amount of gas to cause the explosions. It is possible that they were caused by the sudden oxidation of Na or Na_2O, but the authors were unable, under the conditions, to establish that theory. Although the operators were well protected by screens, operation of the furnace finally became so dangerous that its use was abandoned and the furnace was dismantled. Further work along this line, however, should be of value.

COST OF TREATMENT WITH ELECTRIC FURNACE.

Only a rough approximation of the cost of such treatment can be given, as the work was of purely experimental character. If the explosions could be eliminated, somewhere around 75 pounds of finished product per hour could be obtained with a pressure of about 110 volts and a current of about 600 amperes or about 65 kilowatt-hours. The electrodes wore away at the rate of about 3 inches per 100 pounds of output, and as only about one-half or less of a 30-inch electrode could be utilized, two electrodes would be needed for every 400 to 500 pounds of product.

After the abandonment of the electric furnace, experimental work was continued in an entirely new direction, and finally a method was evolved that appears satisfactory. The details of this method will be described in a later report of the Bureau of Mines.

CHAPTER IV. RADIUM REFINING.

REFINING IN THE PLANT.

The first treatment of the radium-barium sulphates is of great importance, as the whole capacity of the plant, as far as the radium goes, depends upon this first treatment. If the sulphates can not be handled as rapidly as produced, the actual amount of radium refined each year must necessarily be much below the capacity of the plant itself. The tediousness of the process used by Haitinger and Ulrich (see p. 20) can be readily appreciated on reading the account of their work. In order to produce 5 to 10 grams of radium metal per year, it is absolutely necessary to have shorter methods of handling the sulphates than were used in the early days of radium refining.

The first step in this direction is to obtain a high-grade sulphate, as sulphates containing a considerable amount of silica and other impurities are not nearly as readily treated as those that are comparatively pure. Therefore, any process for the treatment of carnotite or pitchblende that involves the precipitation of a high-grade, clean sulphate has a great advantage over other processes involving the precipitation of the sulphate along with silica or allowing the sulphate to remain in the residue with silica and other impurities. This is one of the advantages possessed by the process described in this report. As a rule, the sulphates obtained contain at least 85 per cent $BaSO_4$, and in much of the product the proportion of barium sulphate present is as high as 90 per cent. The proportion of radium in these sulphates has averaged about 1 milligram of radium metal per kilogram of total weight.

OLD METHOD OF TREATING RADIUM-BARIUM SULPHATES.

The old method of boiling the sulphates with a solution of sodium carbonate in order to obtain radium carbonate and sodium sulphate is one that can be applied to any radium barium sulphate. It involves, however, considerable labor and much filtering and washing, as the radium-barium carbonate obtained must be washed completely free from traces of sulphate. The carbonate is then dissolved in chemically pure hydrochloric acid and filtered, and any insoluble material is worked up with the next batch of sulphate. If three parts of sodium carbonate dissolved in water is boiled with one part of radium-barium sulphate, a considerable part of the sulphate is con-

verted into carbonate at one treatment. It is claimed that this conversion can be improved if the heating is done under pressure in an iron tank, although the authors have not tried this method. Any method, however, that will shorten this somewhat tedious process should be welcomed.

RESULTS OF EBLER'S EXPERIMENTS.

The results obtained by Ebler[a] are suggestive. In his experiments he used two sulphates, one low grade and the other high grade. The low-grade sulphate contained 37.1 per cent barium sulphate, 21.8 per cent silica, and 23.9 per cent total sulphate, with traces of vanadium, iron, aluminum, etc. The radium content was only 0.032 milligram per kilogram. The analysis of the high-grade sulphate was as follows:

Results of analysis of high-grade sulphate.

	Per cent.
$BaSO_4$	65.7
$PbSO_4$	4.5
$CaSO_4$	19.3
$FeSO_4$	1.4
SiO_2	5.0
Ra (metal) milligram per kilogram	0.35

On heating these sulphates with carbon to a bright red heat for two to four hours he obtained a 32 per cent reduction with the low-grade and a 68 per cent reduction with the high-grade sulphate.

With calcium carbide the following reaction takes place:

$$MeSO_4+4CaC_2=4CaO+MeS+8C$$

The temperature and time of heating was the same as for carbon. With the low-grade sulphate he obtained an 83 per cent reduction and with the high-grade sulphate a 70 per cent reduction, showing an improvement in carbon with the low grade, but not with the high grade.

With calcium hydride a spontaneous reaction takes place as follows:

$$MeSO_4+4CaH_2=MeS+4CaO+4H_2$$

The reduction with the low-grade material was 75 per cent, and with the high grade 80.5 per cent.

Ebler also tried a mixture of calcium hydride and carbide. The low-grade sulphate required three parts of the former to one of the latter to produce a 64 per cent reduction. The high-grade sulphate gave a 60 per cent reduction with one part of hydride to two parts of carbide.

In general, he found that the crude sulphate was better reduced by the more vigorous reducing agent. Although the reactions with

[a] Ztschr. angew. Chem., vol. 28, pp. 25–48.

calcium hydride and carbide are immediate and spontaneous, the cost and the difficulty of obtaining calcium hydride make the desirability of their use doubtful.

REDUCTION OF BARIUM SULPHATE BY COAL GAS IN AN ELECTRIC FURNACE.

It is a well-known fact that barium sulphate can be reduced by heating in a current of coal gas. This method promised good results, although others had tried it and had reported failure without giving details. As temperature control within reasonably narrow limits was necessary in order to get reduction, and at the same time to get the material in a nonclinkering condition so that it could be readily removed from the tube in which it was heated, the use of an electric furnace seemed more desirable than heating by coal, oil, or gas.

The reduction chamber of the electric furnace consisted of a cast-iron tube 5 feet long and 6 inches in internal diameter. Around this was wound 120 feet of No. 21 nichrome ribbon, the insulation being asbestos cloth and a double layer of mica, the pieces being 4 inches by 5 inches in size.

Reduction begins at 575° to 600° C., the actual working temperature being 725° to 740° C. For raising the temperature a current of 40 to 50 amperes was required, and for maintaining it during the reduction, 15 to 20 amperes from a 220-volt circuit. The gas consumption averaged 200 to 300 cubic feet per run on 25 to 28 pounds of sulphate.

The heat of reduction was positive at high temperatures, producing reaction clinkers, some of which were friable and some glassy and hard to break. The formation of the clinkers did not seem to affect the reduction, which varied from 75 to 90 per cent.

The main advantage of a reduction with coal gas lies in the fact that no foreign material is introduced. The principal disadvantage is the difficulty of maintaining the insulation owing to the deterioration of the mica. Silicon-carbide tubes were also tried in the electric furnace instead of cast-iron tubes without success. As a rule, after 8 or 12 runs a tube would have to be rewound and the insulation renewed, and during reduction the furnace needed constant attention in order to maintain the correct temperature. The iron tube was also gradually attacked by the charge.

A constant-feed furnace of much smaller size, having an alundum tube, was also built. The difficulty was in getting a proper feed through the furnace and a sufficiently high reduction. When the tube got hot it became sticky, and the rate at which the sulphate went through could not be controlled.

REDUCTION WITH CHARCOAL.

While the work was being done with the electric furnace, the reduction of radium-barium sulphates with charcoal was tried. This was done by mixing the sulphate with about one-fifth its weight of powdered charcoal and heating for 7 or 8 hours in a graphite crucible in an oil furnace, the temperature being about 800° C. Reduction takes place readily, according to the equation $BaSO_4 + 4C = BaS + 4CO$, with carbon monoxide burning at the spout of the crucible. When the evolution of gas has stopped, the crucible is allowed to cool with the lid on, and the radium-barium sulphide is removed and broken up. Under such conditions, with a sulphate reasonably free from silica, it is usual to get a 90 per cent reduction of the sulphate to sulphide, the ratio of the reduction of the radium being the same. The sulphide is then dissolvēd in dilute chemically pure hydrochloric acid. This step should be performed out-of-doors or with a hood having a good draft, as the evolution of hydrogen sulphide, of course, is large. The gas also catches fire spontaneously at times, owing to the possible formation of small quantities of phosphides, although it is difficult to see how these could be formed under the conditions. If the work is conducted in open vessels in the open air this danger is minimized. The residue obtained on filtration consists of undecomposed radium barium sulphate, a small amount of charcoal, and some silica. The amount of radium in this residue is usually about the same as that in the original sulphate. The residues are stored and re-treated in the same manner, although filtering them after leaching with acid is more difficult than filtering a sulphide obtained from an original sulphate.

After the re-treatment of these residues, a third residue is, of course, also obtained. There is a tendency for the accumulation of lead in the second and third residues, especially the latter, and, therefore, it is better to fuse the third residue with sodium carbonate than to reduce it once more with carbon in the usual way. After fusion, during which most of the lead is eliminated, the material is thoroughly washed and dissolved in chemically pure hydrochloric acid. As the weight of the second residue is only 1 to 2 per cent of that of the original radium barium sulphate treated, it is necessary to carry out this treatment only once or twice a year.

The reduction with carbon is the method that has been used almost exclusively at the plant of the National Radium Institute. An oil furnace (*23*, Pl. III) holding three No. 100 graphite crucibles handles all of the sulphate made in the plant, the furnace being used only every other day. The actual amount of charcoal required is a little more than one-fifth the weight of the sulphate treated. The charcoal and sulphate are ground in one or two small ball mills (*66*, fig. 2),

a short piece of 1-inch steel rod being placed in each mill instead of balls. This treatment for 10 or 15 minutes is sufficient to intimately mix the sulphate and the charcoal. The ground material is then carefully removed from the ball mill and is placed in the crucibles, which are then transported to the furnace room.

TREATMENT OF SULPHIDE.

After reduction, the sulphide is ground in a similar manner in the other mill and is then leached with moderately dilute chemically pure hydrochloric acid in an earthenware pot placed just outside of the building. No particular precautions are necessary to prevent oxidation of sulphide to sulphate. If such oxidation does take place, the amount of radium and barium precipitated will be proportionate to the relative quantities of these elements present—roughly, 1 part per million. The material in the pot is then transported in crocks to an earthenware suction filter, and the filtrate, which consists of an almost saturated solution of radium-barium chloride, is ready for fractionation. In actual practice it has been customary to use an amount of acid that will not be saturated with barium chloride when hot, but will be almost saturated when cold. This precaution reduces the boiling afterwards required to get the first batch of crystals.

EQUIPMENT FOR FRACTIONATION.

The actual arrangement for fractionation as indicated in figures 2 and 3 is not the arrangement that has been used up to the time of writing (September, 1915), but is what will be used after additional acid-proof ware has been received from France. In the past it has been necessary to use an inadequate equipment, so that the refining capacity has been less than that of the rest of the plant. With the equipment shown in figures 1 and 2, however, it will be possible to refine considerably more than the total output of the plant, now about 400 milligrams per month.

PROCEDURE IN FRACTIONATION.

The method of treatment with this new equipment is to be as follows:

The radium-barium sulphides are dissolved in an earthenware pot, *63* (figs. 2 and 3). The solution and the insoluble material are run onto the upper part of a suction filter, *64*. After filtration and washing, the liquor is run into the upper steam-jacketed vessels, *65*. These vessels are made of silica-lined acid-proof ware, as previously described (p. 44), the upper three each having a capacity of 250 liters and the lower two each having a capacity of 200 liters. The evaporators are covered by two wooden hoods, in the flue of which is placed a

small steam jet to produce a sufficient draft to carry off the hydrogen chloride fumes. The barium chloride solution in the first evaporator, *65*, is then so concentrated that on cooling one-half of the barium chloride crystallizes out. The liquor is siphoned into evaporator *65a* and the process repeated, one-half of the remaining barium chloride being allowed to crystallize. This procedure can be continued all the way down the series, or a fresh batch of liquor may be introduced into *65a* after the crystals have been removed.

The system may be established in several ways, either by adding fresh material to *65* each day and gradually working the mother liquors down the evaporators to the bottom of the series, or by crystallizing the first batch of liquor introduced into *65* all the way down the series before the introduction of fresh material. Whichever method is used, the procedure afterwards adopted is the same. Half of the barium chloride in evaporator *65d* is crystallized out and removed from the mother liquor. This liquor should then be discarded entirely. After the crystals have been removed from the evaporator, the liquor obtained from a similar crystallization in *65c* is siphoned into *65d*. The liquor from a similar crystallization is then siphoned from *65b* into *65c*, and the crystals from *65d* are dissolved with this liquor in *65c*. In other words, the crystals from each evaporator ascend, and the mother liquors at the same time descend, the crystals from a given evaporator being dissolved in the liquor from the second evaporator above.

For example, the crystals from *65d* will be dissolved in the liquor from *65b* in the evaporator *65c*, with the addition of the minimum amount of water required, the crystals from *65c* will meet the liquor from *65a* in *65b*, and will be dissolved in this liquor, and so on up the series. The new material is always introduced into *65*. The crystals obtained from *65* are then refractionated twice in smaller pots (*62*) of about 80-liter capacity, the liquor from the first crystallization coming back into *65* in the same manner as already described. Evaporator *65*, therefore, may be considered as the "zero" pot, with the barium going in one direction and the radium in the other. After the second recrystallization in the smaller pots, the material is ready to go to the laboratory, the ratio of radium to barium varying from 4 to 10 parts per million. The whole method is more fully described on pages 76–86.

The crystallization factor of hydrochloric-acid solution is about 1.5 to 1.6; that is to say, if 50 per cent of the barium chloride is removed, there will be 50 per cent more radium in the crystals removed than in the liquor left behind. Owing to lack of equipment only two "minus" crystallizations have in the past been given in acid solution. The average radium content of the liquor thus obtained has been too

high to discard, averaging during the past few months, 0.0679 milligram of radium per liter of solution, or a ratio of radium to barium of 187.5 parts per billion. With three, or even two more "minus" crystallizations, as planned, the radium content of this material could readily be reduced so that the final liquor could be discarded.

In actual practice up to September, 1915, this solution has been neutralized with ammonia, preferably filtered, and fractionated in the manner already described, in a series of seven bathtubs shown at *52* in Plate VII. The head batch of crystals is then put back at a suitable point in the acid crystallizing system and the tailings either discarded or used as barium chloride solution for the precipitation of fresh batches of barium sulphate in the plant. During the past few months, the average radium content of the tailings discarded, or used as described, has been 29 parts per billion. Ordinarily, material having 40, or even 50, parts of radium per billion might well be discarded.

At all necessary stages in the fractionation, samples are taken for the determination of radium, so that a check may be kept on the progress of the work.

LABORATORY REFINING.

Radium of any desired degree of purity may be obtained by fractional crystallization of the barium solution, first as chloride and later as bromide. The radium continues to be enriched in the crystal fractions and impoverished in the successive mother liquors, as already described.

The salts of most metals, such as iron, aluminum, and vanadium, that may occur with the radium-barium salt as impurities, pass into the mother liquors, and give no difficulty in the enriching radium fractions. Lead is an exception, however, and requires special treatment. Where barium (radium) sulphide, after reduction from sulphate is dissolved in hydrochloric acid solution with copious evolution of hydrogen sulphide (see p. 74), one might expect the almost complete removal of the lead content. This by no means occurs, no matter what precautions are taken in the way of final acid concentration or other conditions, such as have been prescribed by other authors.[a] The treatment necessary to remove all lead and the reasons for its seemingly abnormal action are discussed in a subsequent section.

Figure 5 shows the steps of the concentration system employed in the laboratory.

[a] Ebler, Erich, and Bender, W., Sitzb. Heidelberger Akad. Wiss., Abt. A, Abh. 18, pp. 6, 7; Ztschr. anorg. Chem., vol. 83, 1913, pp. 152-153.

TREATMENT OF CHLORIDES.

The crude barium (radium) chloride received from the plant, containing 4 to 10 parts of radium element per million, is dissolved in water in large porcelain dishes (Pl. XIII, *A*), and hydrochloric acid

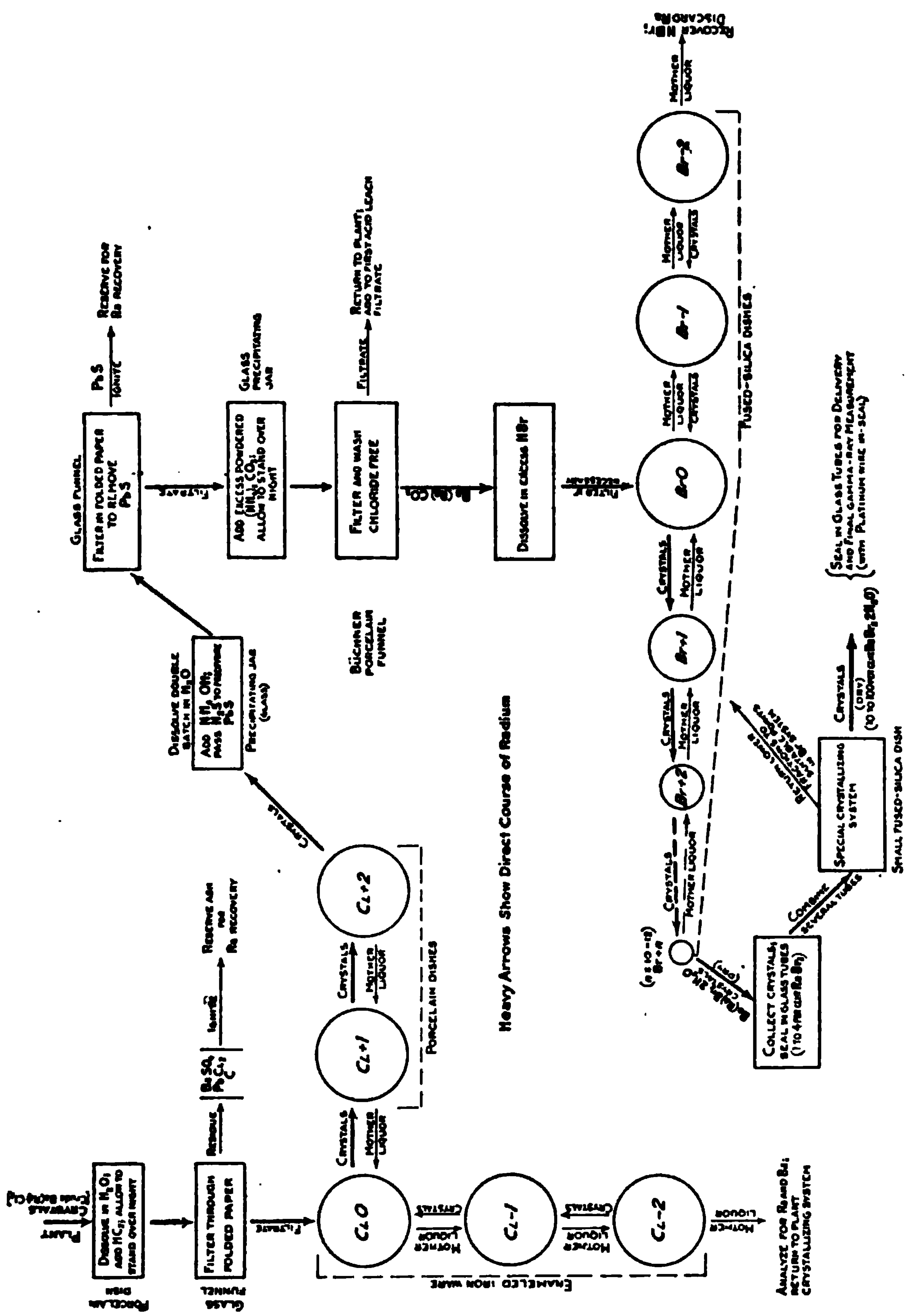

FIGURE 5.—Diagram of laboratory crystallizing system.

is added to precipitate as much lead as possible. After the solution has stood over night, it is filtered to remove lead chloride and whatever carbon and barium sulphate may have escaped previous filtration. For this purpose a Schleicher & Schüll 50-cm. folding filter is used repeatedly (10 to 12 times) until clogged, when it is washed,

dried, and ignited together with the residue, which is held for treatment to recover the radium.

The filter ash and residue, after the passage of about 1,500 milligrams of radium element, weighed 9.5 kilograms and contained 37 milligrams of radium, or about 2.5 per cent of the total put through. The residue would therefore contain about 4 milligrams of radium per kilogram, being considerably richer in radium than the original first sulphates. During part of the operations, however, the residue ran as low as 1 milligram per kilogram (about the same as the original first sulphates), which shows that thorough washing can reduce the radium ratio at this point to that in the original insoluble material.

The filtrate is made up to an 8-liter volume, and, after a sample of 0.01 c. c. has been taken for analysis, the filtrate is introduced into the chloride crystallizing system in the vessel designated *Cl0* in figure 5. The vessels used for *Cl0*, *Cl−1*, and *Cl−2* (Pl. XIII, *A*) are all cast-iron ware lined with silica and were obtained from the Danto-Rogeat Co., of Lyons, France. At the time of writing (September, 1915) they have withstood continuously for 9 months hydrochloric-acid solutions which were boiled for several hours daily and yet show little or no deterioration. The pots contain, in the order named above, 16.6, 13.6, and 10 liters, and each is suited to handling about 2½ to 3 kilograms of barium chloride in one daily crystallization. The pots are mounted on iron-ring tripods and heated with perforated-cap gas burners 5 inches in diameter (see Pl. XIII, *A*).

As the factor of enrichment of radium chloride from barium chloride and also of radium bromide from barium bromide is more favorable in acid than in neutral solutions, a fair acid concentration is maintained throughout the chloride and bromide systems.

As indicated in figure 5 in both systems the crystals moving forward are combined with mother liquors moving backward from pots two places removed from each other in the system. Another advantageous feature of the procedure employed consists in introducing the new material, not into terminal but into intermediate vessels, so that the system may be shortened or lengthened at either end without disturbing the rest of the system. For vessels *+1* and *+2* in the chloride system either porcelain or fused silica vessels may be conveniently used.

The mother liquor from *Cl−2* is collected in 100-liter lots, which are analyzed for radium and barium before being returned to the crystallizing system at the plant. The solutions, which are saturated with barium chloride, contain 0.16 to 0.32 gram of barium chloride per cubic centimeter, the exact quantity depending upon the degree of acidity maintained, and have varied in radium content from

A. LABORATORY CRYSTALLIZING SYSTEM FOR CHLORIDES.

B. LABORATORY CRYSTALLIZING SYSTEM FOR BROMIDES.

0.00001 to 0.00020 gram of radium per liter, the exact content depending on the quantity of radium accumulated in the system and also the acidity. The total quantity of radium thus returned to the plant out of 1,500 milligrams put into the system was 60 milligrams, or about 4 per cent.

The chloride crystals from *Cl+2* are dissolved in water without acid, and the solution from two daily batches is brought into a large glass precipitating jar 16 inches high and 8 inches in diameter and made slightly ammoniacal. Hydrogen sulphide is passed in until the precipitation of lead is complete as lead sulphide. This sulphide is filtered and collected according to the procedure for lead chloride and barium sulphate already described, and analyzed. The amount of lead sulphide thus collected during the treatment of 1,500 milligrams of radium was 700 grams, which contained 3 milligrams of radium or 0.2 per cent of the total radium involved. The lead sulphide is stored for treatment to recover the radium. Some lead sulphide was collected, after sulphuric acid free from lead had been used at the plant, in order to preserve a sample of lead coming exclusively from a uranium mineral.

REMOVAL OF LEAD.

The difficulty of removing lead has already been mentioned. Attempts to precipitate it as sulphide even from slightly acid solution usually result in obtaining an orange-red precipitate, which is a sulphochloride of composition $(PbS)_x.PbCl_2$, and far more soluble in acid than lead sulphide; hence complete precipitation of the lead is possible only in ammoniacal solution. Explanation of this action is to be sought in the high concentration of barium chloride in the solution producing such a high concentration of Cl' ions that the dissociation $PbCl_2 \rightleftarrows PbCl^+ + Cl' \rightleftarrows Pb^{++} + 2Cl'$ is almost entirely arrested at the intermediate step. The absence of Pb^{++} ions and the readiness of Pb_2Cl_2S to form according to the equation $2PbCl^+ + S'' = Pb_2Cl_2S$, explains the impossibility of obtaining lead sulphide. Precipitation of Pb_2Cl_2S is only partial so far as lead is concerned and does not take place unless the acid concentration is low. Complete removal of lead as normal lead sulphide is readily accomplished, however, by passing hydrogen sulphide in ammoniacal solution. The loss of radium here has already been shown to be only 0.2 per cent.

CONVERSION TO BROMIDE.

Into the filtrate from the lead precipitate powdered ammonium carbonate is introduced gradually with vigorous stirring until all the barium has been precipitated as carbonate. Where a slight excess of reagent is added, the whole is allowed to stand over night for the

settling of the barium carbonate and the thorough precipitation of the radium carbonate. The supernatant solution is siphoned off as far as possible, and the rest goes to a 9-inch Büchner funnel, where it is filtered and washed several times with suction. The filtrate, together with the wash water, designated "laboratory carbonate liquor," is stored in 50-liter lots and returned into the acid leach liquor at the plant to recover radium and assist in neutralization. The amount of radium in the liquor is surprisingly low, usually 0.001 to 0.003 milligrams per liter, and out of 1,500 milligrams of radium only 6.4 milligrams, or 0.4 per cent, remained in this liquor. In most lots the amount was much lower than this average.

TREATMENT OF BROMIDES.

By inverting the Büchner funnel the barium (radium) carbonate precipitate is removed, and after being separated from the filter paper is dissolved in chemically pure hydrobromic acid (20 to 35 per cent) in the precipitating jar already used. The Büchner funnel and the filter paper are also washed with 1 to 1 hydrobromic acid. If necessary, this solution is again filtered and washed through a Büchner funnel and then introduced directly into the *Br0* vessel.

All the vessels used in the bromide crystallizing system (Pl. XIII, *B*) are of fused silica ware, which can be more safely heated by free flame than porcelain and the solution in silica ware "crawls" less. For *Br0* the 12-inch or 15-inch dish is convenient; for *Br−1* and *Br−2* the 12-inch dish is sufficient, and the sizes fall off rapidly in the plus direction to the smallest silica dishes obtainable.

The degree of acidity to be maintained both in the chloride and bromide crystallizing systems can not be stated definitely, as it depends on operating conditions and must be left somewhat to the judgment of the operator. In general, it may be stated that the acid concentration should be maintained as high as consistent with the size of the crystal crop to be obtained and the quantity of acid fumes that can be conveniently liberated.

The silica vessels in the bromide system are heated on tripods with bare gas flames until the richer fractions are reached, when the evaporation is carried out on an electrically heated water bath in which only distilled water is used, which is shielded from the heating coils by a block-tin protector. In case of an accidental loss of radium solution into the bath, all the water can be drawn off and returned into the system just before the treatment with hydrogen sulphide.

The evaporation required to obtain a suitable batch of crystals may be generally regulated by concentrating the solution until vigorous fanning just begins to cause the formation of crystals on the surface of the hot solution. Of course, the higher the acid concentration

the more generous the crystal batch will be. All vessels in the chloride system are allowed to cool over night, so that only one crystallization a day is made in each vessel in this system. In the bromide series, in which the vessels are smaller, cooling and crystallization proceed rapidly enough to permit several batches being put through in a day if desired. In warm weather the use of an ice box to contain the vessels of the bromide system facilitates operation. The vessels of the bromide system, when not being heated, are kept in order on a board with holes of sizes suited to the sizes of the various dishes. (Pl. XIII, *B*.)

The barium-chloride crystals invariably form a mat on the bottom of a vessel, from which the mother liquor is drained with some difficulty, whereas the bromide crystals form in heavy individual needles, from which the mother liquor is poured off clean with great ease. Only when the bromide system is disturbed by the presence of lead or chloride salts or when it contains too little acid do the bromide crystals fail to form in their characteristic habit. Both disturbing influences should be carefully avoided, as radium concentration is at once retarded if the bromide crystals do not retain their type.

Owing to the high factor of enrichment as bromide, the radium content of the mother liquor from *Br—2* is extremely low. Out of about 1,400 milligrams of radium treated in the bromide system, only 3 milligrams, or 0.2 per cent, went back into this final bromide liquor. In four out of six lots the radium content was only 20 to 30 parts per billion of salt. As it is not profitable to recover radium at a concentration below 40 parts per billion parts of barium salt, the liquors were treated for the recovery of hydrobromic acid only. In the two lots exceeding this concentration the barium-radium sulphate obtained in the acid recovery was stored, to be handled as second sulphates.

The method of recovering hydrobromic acid from the final bromide liquor just mentioned consists in adding a slight excess of sulphuric acid to precipitate all barium, then in adding a slight excess of the original barium-bromide liquor to remove any excess of sulphate, filtering on a large earthenware Büchner funnel, and distilling off hydrobromic acid from the filtrate, a large round-bottomed glass flask and a glass condenser being used. The distillation is repeated until at least a 20 per cent acid is obtained.

NUMBER OF FRACTIONS EMPLOYED.

The number of fractions employed in the plus direction in the bromide system varies somewhat with conditions, 10 to 12 being the usual number. The crystallization is conducted in such a way that

the barium (radium) bromide collected in the final fraction should not fall below 1 per cent of radium bromide, and sometimes is as high as 4 per cent. The total weight of the fraction should be 1 to 2 grams. The factor of concentration for each step in the acid bromide system is about 2 to 2.2, and in the acid chloride system about 1.5 to 1.6.

The procedure in collecting the final fraction is as follows: The mother liquor having been poured back, the crystals are dried in the silica crystallizing dish, first on the steam bath, and finally in an air-drying oven at 105° to 110° C. until thoroughly dry. The crystals are then collected in a previously-weighed glass tube sealed at one end, 4 to 6 millimeters in internal diameter, and of sufficient length to contain the salt and permit the other end to be sealed with the blast. The total weight of salt is determined by weighing the sealed tube and contents together with the detached end. The salt is kept stored in the tube, and gamma-ray measurements of the radium content are made at intervals of a few days until the determination is thoroughly established. When a number of tubes have been collected, they are opened and the contents combined for further fractionation.

In opening tubes that have been closed for several weeks, care must be taken that the contents shall not be scattered by the release of gas that may have been generated by the chemical action of the alpha rays, especially if the salt was not thoroughly dried when placed in the tubes. Two precautions may be taken—one end may be drawn out to a capillary, which may be opened first, thus relieving the pressure; or a scratch may be made near one end, which may then be inserted well into a beaker lying horizontally and the tube opened by a hot piece of glass applied to the scratch. If a capillary end is provided, it is preferable to have this so narrow that the salt crystals can not enter and the salt will remain confined in a definite length of tube, one not too great for accurate gamma-ray measurements. It is also desirable that the drying and sealing of a tube shall follow crystallization as promptly as possible in order to furnish a definite zero point from which the accumulation of gamma radiation may be dated. This detail is more fully discussed in a subsequent description of measurements (p. 89).

PROCEDURE IN HIGHER FRACTIONATION.

After the collection and measurement of the salt in a sufficient number of tubes (usually 10 to 20), the higher fractionation may be commenced. If the percentage of radium is approximately the same in all tubes, they may be combined into one solution; if not, two or three solutions are made, which later fall into the crystallization series in their proper places.

As a guide for procedure in making this special fractionation, it has been found convenient to assume a concentration factor of 2.0 for the radium enrichment, which means briefly that if one-half of the salt is crystallized from a given fraction, two-thirds of the radium is contained in the crystals and one-third in the mother liquor. The advance fractions are then made and the lower ones recombined as suggested by this factor until it is thought that the desired products have been attained, when the crystals are collected and sealed as already described for the lower-grade salt. If the products prove on measurement not to have the desired percentage of radium, the tubes are opened again and either the salts are suitably recombined, or possibly further fractionation is carried out.

As regards the higher grade salts, one additional precaution is observed in sealing the tubes, namely, a small platinum wire is sealed through one end of the tube to conduct away the unipolar charge that may collect in the interior, attaining voltages that could cause destructive sparking. Reports are on record of serious radium losses having resulted through neglect of this precaution.

RESULTS OF HIGHER FRACTIONATION.

The following data from a series of higher fractionations will serve to illustrate the procedure:

Results of recrystallization of radium-barium bromide.

BEFORE RECRYSTALLIZATION.

Tube No.	Weight of salt.	Ra.	$RaBr_2 2H_2O$.
	Grams.	*Milligrams.*	*Per cent.*
31	1.1705	19.9	3.16
32	1.0873	15.5	2.65
33	1.5074	43.0	5.30
34	1.4178	31.6	4.15
35	1.6325	45.0	5.13
[a] 35	1.6355	21.6	2.46
36	1.9548	23.3	2.21
37	1.4545	21.8	2.90
	11.8603	221.7	[b] 3.48

AFTER RECRYSTALLIZATION.

C-1	0.1630	76.7	87.8
C-2	1.0983	107.3	18.2
C-3	.8732	31.7	6.7
C-4	.7101	4.17	1.1
C-5	1.0265	.62	.11
C-6	1.7256	1.56	.16
C-7	3.5449	.43	.02
C-8	2.0398	.12	.01
C-9	.4542	.07	....
	11.6356	222.67	....

[a] Special. [b] Average.

As will be noted, the total weight of salt at the start was 11.86 grams, and after recrystallization 11.64 grams. The difference was due to material adhering to crystallizing dishes in the poorer fractions. The total Ra at the start was 221.7 milligrams, and after recrystallization 222.67 milligrams. The difference was due to a difference in measurements.

Tubes C–1, C–2, and C–3 were the fractions preserved, the others were returned into the crystallizing system at suitable points.

COMMENTS ON RESULTS.

In beginning this series, tubes 33, 34, and 35 were combined and recrystallized in twelve "head" fractions directly to tube C–1. To the first mother liquor from tubes 33, 34, and 35 were added the contents of the other tubes, Nos. 31, 32, 36, and 37. This series was carried in about 15 recrystallizations up to tube C–2. The other lower fractions in the C series were obtained by further fractionation and combination of the mother liquors, as suggested by the factor 2.0.

In general, the fractionation of radium bromide from barium bromide by means of crystallization is simple and sure. No especial skill of the operator is required, as many authors lead one to believe, nor does it seem possible that the substitution of other processes, such as the adsorption of radium by means of colloidal gels or the fractionation of radium in concentrated alkaline solution would prove advantageous. The applicability of such methods should be regarded as doubtful until demonstrated on an actual working scale. There is a wide difference between commercial operation and the extraction of a small quantity of radium in the laboratory.

The time of only one man continuously is required to carry on the laboratory refining through the system indicated in figure 5, and the time of one additional man is required for four to six days each time the higher fractionation is performed.

ADSORPTION OF RADIUM.

Contrary to rather widely entertained views, the production of a precipitate, or the presence of one, in a radium-barium solution does not necessarily involve the adsorption of considerable quantities of radium from solution. As has already been pointed out, the precipitation of 700 grams of lead sulphide in ammoniacal solution carried with it only 3 out of 1,500 milligrams of radium. The recent principle of adsorption established by the work of Paneth [a] points out that any serious removal of radium should be expected only in case the radium forms an insoluble salt with the negative radical of the adsorbent. Without entering at present into discussion as to whether

[a] Physik. Ztschr., vol. 15, 1914, pp. 924–929; Ztschr. physik. Chem., vol. 89, 1915, pp. 513–528.

this removal is really due to adsorption or to ordinary chemical precipitation, it is interesting to note that the failure of radium to be removed as sulphide, together with lead, accords well with its chemical nature, as one would expect radium sulphide to be soluble. On the other hand, the completeness of its removal as carbonate, which is to be expected, has been shown by the fact that only 6.4 out of 1,500 milligrams of radium element passed into 300 liters of filtrate.

Even if barium sulphate is present in or is precipitated in a solution containing radium, the removal of the radium is small provided a large excess of barium is present. A discussion of the "protective" action of barium appears in the section on measurements (p. 92).

ACCUMULATION OF RADIUM IN LABORATORY CRYSTALLIZING SYSTEM.

It is desirable to allow several hundred milligrams of radium to accumulate in the crystallizing system. This accumulation tends to increase gradually unless special preventive means are taken, such as crystallizing out special fractions without corresponding introduction of fresh material. The following table serves to show the results of operating the system during a period of about one month:

Third balance sheet for radium in laboratory-crystallizing system.

Tube No.	Date.	Weight of radium introduced.	Total weight of salt withdrawn.	Weight of radium withdrawn.	Balance of radium in system.
	1915.	*Milligrams.*	*Grams.*	*Milligrams.*	*Milligrams.*
Balance after delivery of radium on Apr. 1..........					271.8
38..	Mar. 12.. Mar. 16..	15.3 16.7	1.463	26.7	277.1
39..	Mar. 13.. Mar. 15..	11.2 25.9	2.103	41.5	272.7
40..	Mar. 17.. Mar. 18..	22.8 19.4	2.751	25.2	289.7
41..	Mar. 20.. Mar. 22..	18.5 16.8	.9915	20.4	304.6
42..	Mar. 23.. Mar. 24..	14.3 17.6	1.551	17.7	318.8
43..	Mar. 25.. Mar. 26..	16.5 19.5	1.100	19.8	335.0
44..	Mar. 27.. Mar. 29..	20.0 9.9	1.094	22.4	342.5
45..	Apr. 1... Apr. 2...	12.1 15.3	1.759	30.8	339.1
46..	Apr. 3... Apr. 5...	12.7 10.5	1.631	41.7	320.6
47..	Apr. 6... Apr. 7...	8.5 11.1	.938	23.8	316.4
48..	Apr. 8... Apr. 9...	11.5 8.6	1.467	33.0	303.5

Changes in the richness of the deliveries into the system affect concentration in the higher fractions with a certain lag. The distribution of radium in the different vessels of the system is somewhat accidental, dependent on the size of fractions taken, acidity, and other conditions that may vary from time to time, but in general, radium tends to accumulate in the −1, 0, and +1 fractions both in

the chloride and bromide systems. The total accumulation in the bromide is greater than in the chloride system on account of its greater number of fractions, the higher solubility of barium bromide, and the higher degree of concentration of the radium.

All the refining operations described in this section are carried out in a room 12 by 20 feet. The vessels of the chloride system are handled over a large, shallow copper trough to prevent possible loss of solution by "crawling," possible breaks, or other accidents. Incidentally, it may be mentioned that the refining work up to the present has been notably free from misfortunes, and that no serious losses of radium have ever occurred. Although a hood is not necessary, it would sometimes prove advantageous in carrying off acid fumes. Usually, however, solutions are not sufficiently concentrated for the acid fumes to become very objectionable.

CHAPTER V. RADIUM MEASUREMENTS.

One of the most essential factors in the successful production and concentration of radium consists in following the material being concentrated, by means of careful quantitative determinations, through all the various operations from the original ore to the final product. This involves the radioactive analysis of a large number of products differing widely in chemical and physical properties and also varying in radium content through more than a billion fold.

As is well known, elements possessing radioactivity can be detected and even quantitatively measured in quantities far below the limits of any methods based on other properties. This fortunate property more than any other makes possible the concentration of an element from an ore containing about 1 part in 200 millions, by weight, to a product of any desired purity, up to 100 per cent, with a total loss not exceeding 15 per cent.

In the course of the work described in this bulletin, the principles of existing methods of radioactive measurement have been employed, but both instruments and methods have been modified to meet the practical requirements of plant control. One of the first principles recognized is that there is no universal method of radium determination at present and that each product requires study and individual treatment suited to its own peculiar chemical and physical characteristics. Nevertheless effort has been made as far as possible to unify and standardize the methods employed. Simplicity of procedure as far as consistent with accuracy has been the object sought, and the methods herein described are the result of large numbers of determinations made during more than a year of plant operation.

Three general methods of radium measurement have been employed: An alpha-ray method suitable for solids of low radium content, where no great degree of accuracy is desired; a gamma-ray method suitable for solids with a comparatively high radium content, where accuracy is desired; and an emanation method, suitable to any substance from which the radium emanation can be quantitatively liberated, and in the use of which an accuracy of 1 to 2 per cent is obtainable.

APPROXIMATE METHOD FOR SOLIDS.

A simple method of obtaining an approximate idea of the radium content of a solid susbtance not too high in activity consists in comparing its surface radiation with that of a standard substance.[a]

[a] See Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium: Bull. 70. Bureau of Mines, 1914, pp. 64-65.

The activity thus measured is essentially that of the alpha or non-penetrating rays. Although the result obtained may be accurate so far as the surface radiation is concerned, there is evidently no certainty that this indication closely approximates the radium content of the solid. For some ores, especially those ores of closely similar origin and general character, the comparison furnishes satisfactory results, but with other ores the deviation becomes great, on account of the differences in the nature of the gangue material, the radium distribution, the "emanating power," and other variable factors.

Consequently it is evident that the method, although possessing the advantage of simplicity, has no great degree of accuracy.

Actually it has been used in the present work in two instances only—for control of the ore sorting at the mines and for the examination of the residue after nitric-acid extraction of the ore in the plant. As mentioned subsequently, the tailings, on account of their low radium and high silica content, present great difficulty of treatment by the emanation method. For this reason, and also because the low percentage of radium in the tailings allows a considerable relative deviation without invalidating the absolute results, it has been found in most cases convenient to use the solid method in testing the tailings for radium. The determinations have, however, been checked occasionally against those obtained by the emanation method.

The procedure consists simply in comparing the activity of the tailings, after they have been allowed to dry for a few days, with the activity of the original ore, both tailings and ore being spread over the same area on a plate introduced directly into the soild electroscope.[a]

GAMMA-RAY MEASUREMENT OF RADIUM.

The most accurate method for the determination of radium in salts containing an absolute quantity of not less than 0.1 milligram of radium, and having a concentration of at least 0.05 per cent is the gamma-ray method. This method involves comparing the rate of electroscopic discharge produced by the gamma radiation from a standard salt containing a known amount of radium with the gamma radiation of an unknown salt, the conditions of measurement being, of course, identical.

A standard tube should have its radium content determined by a careful comparison with the international standard or one of the subsidiary standards. The Federal Bureau of Standards possesses a standard tube containing 20.28 milligrams of radium chloride, or 15.44 milligrams of radium element, by comparison with which the standard

[a] See Moore, R. B., and Kithil, K. L., loc. cit.

tube of the National Radium Institute measured 10.56 milligrams of radium element.

If the thickness of glass of the containing tube differs from that of the standard, a correction of about 1 per cent per 1 millimeter difference of thickness should be applied. In accurate work a correction for the length of tube should also be made, which will vary with the distance from the electroscope. This correction may be determined in various ways. It has been found convenient for the purposes of the Denver laboratory to approximate the correction by shifting the standard from its normal position until one end coincides with the position that the end of the unknown would take. The percentage decrease in activity owing to slightly increased distance of the ends from the instrument thus determined, may be applied as a positive correction. Or, if preferred, the correction may be determined once for all for tubes of various lengths containing the same quantity of radium at several distances from the instrument.

The precautions to be used in sealing the salts in glass tubes have already been described (pp. 82–83). Only after a given tube has been sealed for one month or more is its gamma radiation proportional to its radium content. In making measurements of tubes sealed for a shorter time, account must be taken of the percentage rate of accumulation of emanation (and consequently the gamma radiation) according to the expression $I_t = 1 - e^{-\lambda t}$, in which I_t is the percentage accumulated at any time, t, e is the base of the Naperian logarithms, λ is the decay constant of radium emanation $= 0.0075$ (hour)$^{-1}$. The function $e^{-\lambda t}$ for various time intervals is solved in the Kolowrat Table A.[a]

If the gamma-ray measurement must be made within a few days after the tube has been sealed, the importance of a sharply defined starting point for the accumulation can be appreciated (see p. 82). If the assumed starting point is correct, two separate measurements at different time intervals will be in accord; if not, a new theoretical starting point must be determined as follows: The percentage increase in activity between the first and second measurements is determined. Clearly there is only one period during which this increase can take place in the given interval of time, which can be found by consulting the Kolowrat table. This figure establishes the the corrected zero point, from which the two time intervals are reckoned anew and applied to the radium measurement. A third measurement at yet another interval will serve to test the correctness of the new zero.

Measurements made in this way, after only two days of accumulation, have agreed within 1 per cent with the measurements made by

[a] Kolowrat, Léon, Le Radium, vol. 6, 1910, pp. 195–196; also Curie, Mme., Traité de Radioactivité vol. 2, pp. 361–362; and Chemiker Kalender, 1914, vol. 2, pp. 361–362.

the Bureau of Standards after a much longer lapse of time. The final accuracy of the gamma-ray method is considered to be 0.3 per cent.

Almost any form of electroscope can be employed for the gamma-ray measurement by placing between the electroscope and the source of radiation a lead screen one-eighth to one-fourth inch thick. The screen should preferably be nearer to the electroscope than to the radium tube. The interchangeable electroscope (see p. 99) with an ordinary emanation chamber has been used satisfactorily.

However, for the sake of convenience, a special gamma-ray instrument has been designed by the authors for these measurements (see Pl. XIV, *A*). It consists of a cylindrical discharge chamber, also containing the leaf system, mounted on a wooden base about 3½ feet long. The telescope is fixed directly into the ionization chamber, which is provided with a small window in the opposite side for the transmission of light. The cylinder is of brass with an interior lead lining one-eighth of an inch thick, besides which an additional lead plate (one-fourth of an inch thick) is provided outside the chamber, which may be removed if desired. The holder for the radium tube is mounted on a track running the full length of the 3-foot graduated extension, and along which it may be fixed at any point by means of a set screw. The tube holder itself is a grooved metal rod, in which the tube is held in a horizontal position opposite the middle of the discharge chamber by a single narrow spring clamp. The groove is graduated from its center in both directions to facilitate centering the tube and making length corrections.

The electroscopic procedure, with respect to charging before use, and determination of the natural leak, is identical with that described in detail for the emanation method (see p. 103).

In general, the gamma-ray method is simple and satisfactory. Its accuracy may be judged from the data contained in the table on page 83, in which all the radium data reported were obtained through its use. As carnotite is free from thorium, no complications arise from the presence of mesothorium in the radium salts.

RADIUM DETERMINATION BY EMANATION METHOD.

The determination of radium by the emanation method involves separating radium emanation (as a gas) from its parent radium, and measuring its quantity in a gas-tight electroscope previously standardized with a known amount of radium emanation. Analyzed pitchblende has been employed to furnish known quantities of emanation for purposes of standardization.

Three general methods of procedure may be used as follows:

(1) Release and measure the emanation from a substance in which it is in equilibrium with the radium content. This condition will

A. ELECTROSCOPE FOR MAKING GAMMA-RAY MEASUREMENTS.

B. ELECTROSCOPE WITH INTERCHANGEABLE IONIZATION CHAMBER. THE ELECTRIC LIGHT SOCKET AND WIRE ABOVE THE INSTRUMENT ARE NOT PART OF IT.

usually not be fulfilled unless the substance has been retained for a month or more in a closed container. In exceptional instances, however, the radium might be contained in a solid of such compact structure, or with a glazed surface, so that no spontaneous loss of emanation could take place. But even with a dense mineral like pitchblende, the leak of emanation, called "emanating power," amounts at ordinary temperature to several per cent. This circumstance suggests the second procedure.

(2) Liberate and measure the emanation retained in the solid and apply as correction the "emanating power," which must be determined separately and preferably after the solid has been in a closed retainer for one month.

Both of the above procedures, applicable in general to solids only, involve in practice long delays, and, although they are adapted to scientific investigation, they are not suited to radium measurement for purposes of plant control when quick results are desired.[a] The following procedure is shorter and probably preferable when its use is possible.

(3) Remove the emanation completely from a sample of the substance to be analyzed for radium, close it at once in a gas-tight vessel, and allow the emanation to accumulate for a convenient period (one to ten days). Then remove it and measure it, making a time correction to find the maximum amount that would have been formed on the attainment of equilibrium.

For removal of emanation the radium must be contained either in solution or in a state of fusion.

Some substances, like carnotite, can be deëmanated merely by heating to a high temperature, but carnotite can not be deëmanated a second time in this way, as the first heating changes its physical state so that a second heating does not produce complete deëmanation; hence heating can be used only in the month-accumulation method.

The removal of emanation from a solution may be accomplished by aspiration or preferably by boiling. Only the latter has been used by the authors and is described subsequently. Removal from a fusion may be accomplished by passing air or some other gas over the fusion, but it is preferable to bubble air through the fused mass or to produce in it an evolution of gas to insure the complete removal of radium emanation.

PROCEDURE WHEN RADIUM IS IN SOLUTION OR IN A SOLUBLE FORM.

The determination of radium in a solution can generally be conveniently carried out directly if a few precautions are carefully observed. It has been repeatedly noted that radium solutions show

[a] For further details of the practice in the use of methods 1 and 2, see Lind, S. C., and Whittemore, C. F., The radium-uranium ratio in carnotites: Tech. Paper 88, Bureau of Mines, 1915, pp. 10-20.

some tendency to lose radium from a solution on standing, a phenomenon that manifests itself in a decrease of the successive quantities of emanation that can be obtained from the solution. This tendency of radium solutions has resulted in the almost complete abandonment of the practice of preserving the solutions over long periods of time for standardization purposes, a practice that produced serious errors in some earlier work. The loss is to be attributed to precipitation or adsorption in a form that will not readily give up its emanation. In general, the presence of precipitates or suspensions in the solution should be avoided, though this source of error has frequently been exaggerated.

Loss of radium by precipitation through small amounts of sulphate, originating either in the glass walls of the containers or in the reagents, should be guarded against. For analytical purposes a suitable preventive measure is adding a large excess of "protective barium." On account of the chemical similarity of barium and radium any precipitant that affects the radium will be removed by the large excess of barium, or, rather, radium and barium will be precipitated in the same proportion in which they occur in solution, and therefore only a minimal quantity of radium is removed. Furthermore, it is desirable that the solution contain nitric acid up to the solubility limit of barium nitrate, which is rather low in the presence of nitric acid of 50 per cent strength. The object of the nitric acid is to prevent the removal of radium as basic salt, which may be formed in neutral chloride or bromide solutions by the action of the alpha particles; hot concentrated nitric acid also has the well-known property of rendering barium (radium) sulphate more soluble (see p. 28).

The two essentials in handling radium solutions for analytical purposes consist, then, in maintaining an excess of barium and a fairly high concentration of nitric acid. Following are described treatments for three kinds of solutions, under one of which any given solution will be included. The reasons for the prescribed treatment will be clear from the foregoing discussion.

TREATMENT FOR SOLUTION CONTAINING BARIUM IN LARGE EXCESS OVER RADIUM.

For a solution containing barium in large excess over radium the treatment is as follows:

Place a suitable portion of the solution—such as will contain about 1×10^{-8} gram of radium—in a small Jena flask, and add to it a suitable quantity of 1:1 nitric acid. Add a few glass beads, and boil 5 to 10 minutes to remove all emanation. Allow slight cooling and then close the flask tightly with a one-hole rubber stopper provided with a glass tube drawn out above to a capillary tip. Seal the tip

while some steam is still in the flask, in order to provide a partial vacuum, which should be maintained until the flask is again opened, thus affording a proof that no outward leak of gas has taken place. Note the exact time and date of sealing.

TREATMENT FOR SOLUTIONS CONTAINING LITTLE OR NO BARIUM.

The treatment for a solution containing little or no barium is to add a suitable portion to 1:1 nitric acid which is saturated with barium nitrate, and to proceed as in the treatment described above.

TREATMENT FOR SOLUTION CONTAINING NO BARIUM BUT AN EXCESS OF A BARIUM PRECIPITANT.

A solution that contains no barium but an excess of barium precipitant, such as sulphate or carbonate is usually a filtrate from a radium-barium precipitation, and requires especially careful treatment; otherwise highly erroneous results will be obtained. If such a solution were boiled off and sealed directly, the results would usually be low, as much as tenfold, and the solution would continue to decrease in emanating power the longer it remained standing. This behavior has led the authors to the belief that in the precipitation of radium in low concentration, or at any rate, its removal, whatever the process, is a progressive time reaction. On the other hand, unduly high results may be obtained, especially in using the correct procedure, if the sampling has been incorrect, for example, when too much of the fine (frequently invisible) precipitate relatively rich in radium has been obtained in a given fraction of the liquor.

Such a relatively rich fraction might easily be obtained in siphoning the liquid from above a sulphate precipitate, if the sample of liduid were taken near the end of the process. It has been found necessary to take samples at intervals during the entire siphoning process and to make a composite solution.

PROCEDURE FOR LIQUIDS CONTAINING EXCESS OF SULPHATE OR CARBONATE.

The detailed procedure for treating a liquid containing an excess of sulphate or carbonate, but no barium, is as follows: An excess of barium salt is added to the liquid, and the precipitate is filtered off. The filtrate containing an excess of barium is made acid with nitric acid to the point of precipitation, and is given the treatment outlined for a solution containing barium in large excess over radium. The precipitate, if barium sulphate, is fused with four to five times its weight of a fusion mixture (1:1 Na_2CO_3 and K_2CO_3), and is treated as described later for fusions. If the precipitate is barium carbonate, it is dissolved in nitric acid containing sufficient sulphuric acid to pre-

cipitate an amount of barium sulphate convenient for fusion, which is filtered off. The filtrate that is obtained may be combined with the original filtrate, and given the treatment as described for a solution containing barium in large excess over radium. All radium is then contained either in the filtrate with excess of barium or in the fused precipitate. Both of these fractions are closed simultaneously (within 15 minutes), so that the time of accumulation will be the same for both lots of emanation, which can be later introduced into one electroscope to determine the total radium.

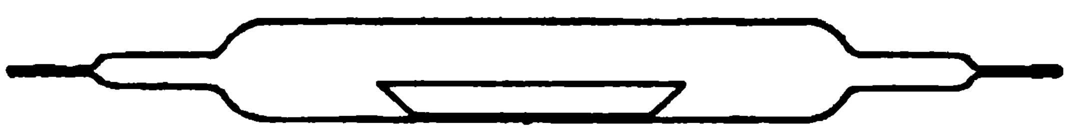

FIGURE 6.—Laboratory apparatus for fusing radium-barium sulphate.

FUSION METHOD FOR RADIUM DETERMINATION.

If the radium is contained in a substance not readily soluble, such as a radium-barium sulphate, fuse a suitable quantity in a small platinum or porcelain boat with four to five times the weight of sodium or potassium carbonate, and note the exact time of cooling. Close this boat in a glass tube as shown in figure 6. Allow the emanation to accumulate two or more days. Connect the glass tube at one end to a highly exhausted electroscope and at the other end to a stopcock. Break the glass tips inside the rubber connection, and exhaust the air from the glass tube into the electroscope several times, leaving enough vacuum in the electroscope chamber to accommodate the gas to be introduced later. Break the glass tube, remove the boat and its contents, wrap in a filter paper, and place in the neck of a flask as shown in figure 7; it is then ready for treatment with 1:1 nitric acid after the flask has been connected with the gas burette, as shown in figure 8. In this treatment the flask is tipped until the acid comes in contact with the carbonate fusion, thus beginning a gas evolution. The stopcock is immediately opened to the gas burette above and the boat and contents are then thoroughly wet with acid and jarred down from the neck of the flask into the body of the acid. As regards larger fusions, the evolution of carbon dioxide may become rapid and care should be taken in handling them, but in small fusions not exceeding 1 gram the boat may be shaken directly down into the acid, which should be heated

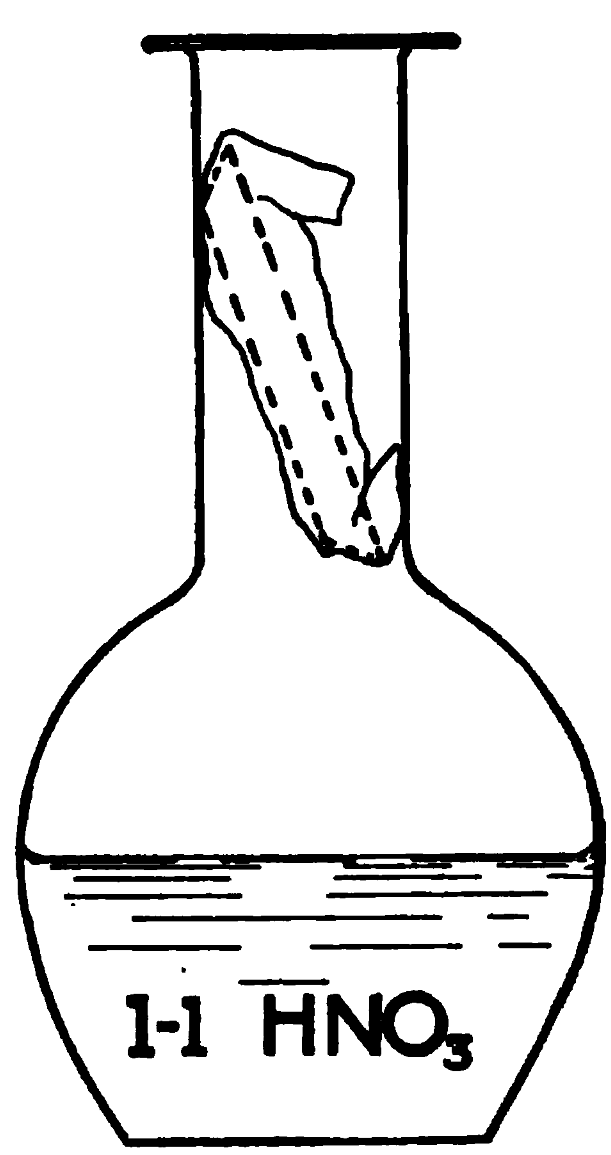

FIGURE 7.—Carbonate fusion in flask for nitric acid treatment.

to boiling as soon as the gas evolution begins to slacken. All of the carbon dioxide is, of course, absorbed by the sodium hydroxide solution in the gas burette. The boiling off from this point on is performed as with solutions discussed below.

For small fusions of substances running about 1 part of radium per million, such as crude radium-barium sulphate and high-grade pitchblende, of which a sample of 20 to 40 milligrams would be taken, the authors have employed small handmade boats, each one being folded from a strip of platinum foil $\frac{1}{1000}$ inch thick, $\frac{3}{4}$ inch broad, and $1\frac{1}{2}$ inches long, the finished boat being about 1 inch long and about $\frac{1}{4}$ inch in cross section. Such boats have been found convenient, can be made at small expense, and have a reasonably long life if the material to be fused does not contain lead. For substances poorer in radium, necessitating larger samples, the authors have employed porcelain boats, flasks holding as much as a liter having been used for the solution in some instances. The gas evolution is so vigorous that a gas burette with an enlarged bulb at the top should be used to furnish an increased amount of sodium hydroxide solution.

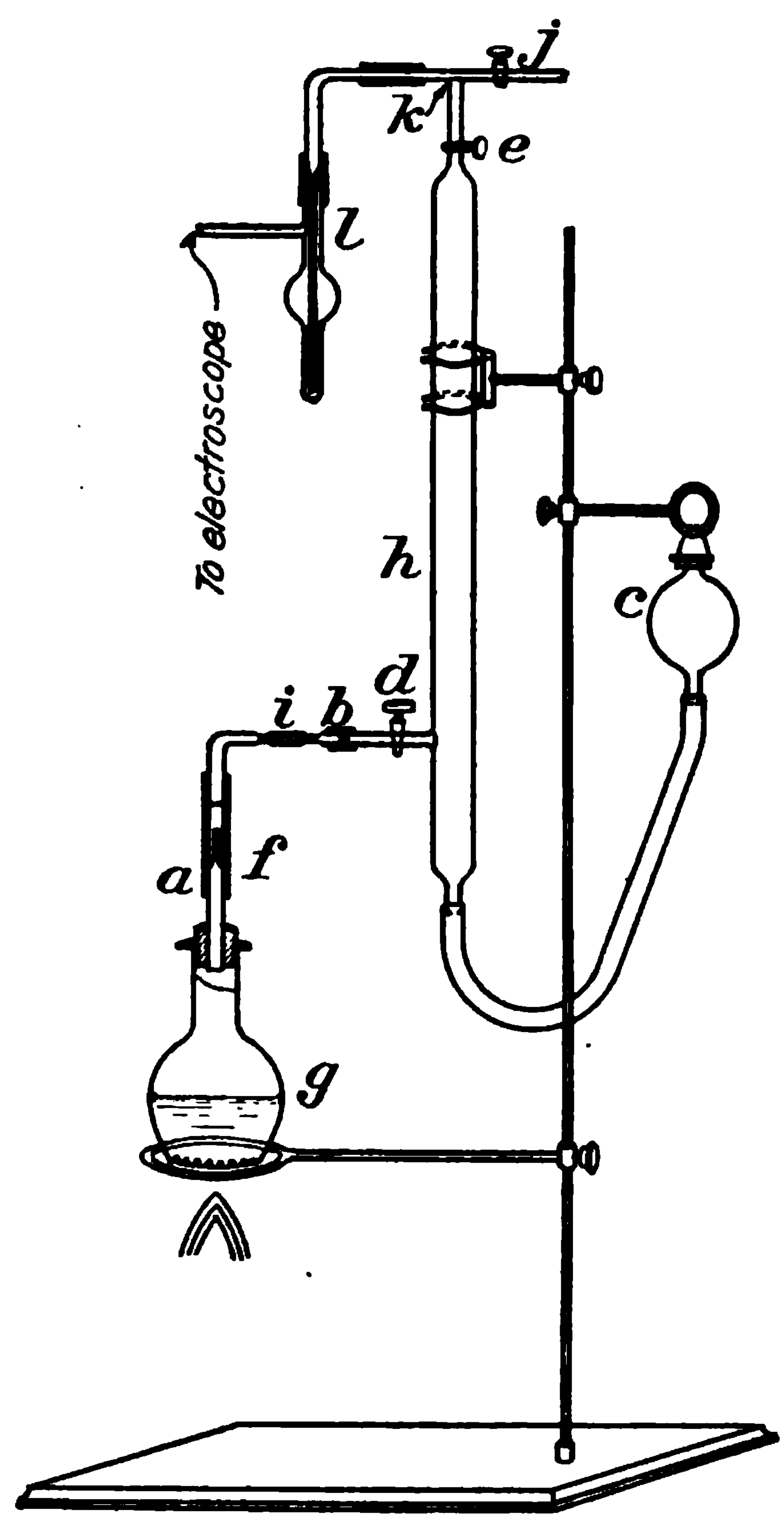

FIGURE 8.—Apparatus for boiling off, collecting, and transferring emanation into electroscope.

DIRECT FUSION METHOD.

If desired, one can use a fusion both before and after the accumulation of emanation instead of dissolving the fusion in acid. If this is done, as soon as the initial fusion cools, the thin platinum boat is unfolded, and the fusion is put into a Jena glass tube of the form shown in figure 6, and held in place at both ends by small glass-wool plugs, which react with the carbonate in the second fusion, giving an evolution of carbon dioxide, which assists in removing the emanation. Usually in this method the gas is not passed into a gas burette at all, but the exhausted electroscope is attached and allowed to pull a current of air directly through the

hard-glass tube while it is being strongly heated with a Meker burner until the vacuum is exhausted. However, this practice would not be allowable except with substances free from thorium.

The tube is heated until it collapses completely, but collapse should not occur until a large volume of air has passed over the fusion. A small drying bulb is placed in front of and another is placed behind the tube. Between the hard-glass tube and the drying bulb next to the electroscope a small tube containing potash solution is introduced to prevent carbon dioxide from passing into the electroscope.

No gas except air should ever be introduced into the electroscope with the emanation because the specific ionization of different gases differs from that of air, and the difference may cause a large error in the comparison.

BOILING OFF EMANATION FROM SOLUTIONS.

For boiling off emanation from solutions, the procedure prescribed is as follows:

Set up apparatus as shown in figure 8, wiring the rubber connections at *a* and *b* to insure tightness. Put into the leveling bulb *c* a stick of sodium hydroxide 2 to 3 inches long, or more if a large quantity of carbon dioxide is to be absorbed; make sure that stopcock *d* is closed and stopcock *e* open; pour boiling distilled water into the leveling bulb and allow the alkali to go into solution. If the boiling is too violent, put a one-hole stopper lightly into the mouth of the leveling bulb. After the alkali has gone into solution raise the leveling bulb until the gas burette is filled to the stopcock *e*. If the quantity of air to be boiled off is small, some air may at first be left in the gas burette. Close stopcock *e* and lower bulb *c* to its original position. Break the glass tip *f* inside the rubber tubing at *a*, and slowly open *d* to ascertain whether there is vacuum in the flask *g*. If so, close *d* again and begin to heat flask *g* over wire gauze. Test the vacuum every few seconds and as soon as the pressure is outward open *d*, and cause the flask to boil vigorously. Continue boiling until live steam has heated to boiling all the liquid in the gas burette *h*. This boiling should never be less than 5 minutes, and sometimes 10 to 15 minutes' boiling is desirable.

After the glass tip *f* has been broken the liquid is likely to be carried upward by steam and in some instances has lodged in the stopcock *d* and caused serious explosions. As a precaution, a roll of thin platinum foil can be introduced into the glass tubing, as indicated at *i*, or the stopcock *d* may have a wide bore, which also obviates the danger mentioned.

After the boiling off has been completed, remove the flame, and as soon as the liquid begins to draw back through the stopcock *d* close the stopcock and remove the flask entirely. Evacuate the electroscope chamber to a suitable vacuum, either by means of an aspirator or, more conveniently, a hand pump; connect the sulphuric-acid microdrying bulb *l* to the electroscope and to the gas burette, as indicated in figure 8. Be sure that stopcock *j* is closed; open first the cock of the electroscope for a moment and reclose it; then slowly open stopcock *e* to full width, and then gradually open the stopcock to the electroscope, allowing the gas to bubble through the microdrying bulb at a fairly rapid rate. When the liquid in the gas burette has risen exactly to the point *k*, close stopcock *e* and open stopcock *j*, allowing dry, dust-free air, which should preferably be taken from outside the laboratory, to sweep out the connections for a few minutes; then close the stopcock to the electroscope, reopen stopcock *e*, and allow the liquid in the gas burette to fall back 3 or 4 inches below the shoulder; close *e*, and then pour off all excess liquid out of *c*; close *j* and again open *e* to the electroscope, allowing air to bubble from the bottom of the gas burette *h* through its entire length to insure the removal of any emanation that may have remained dissolved in the liquid. This precaution is perhaps unnecessary, as the hot sodium hydroxide solution certainly does not take up much emanation, but nevertheless the precaution is in the direction of accuracy. Air should be allowed to bubble into the electroscope chamber until normal pressure has been almost restored.

The procedure just described for boiling off radium emanation is used for carbonate fusions introduced into acid, and also in handling any solids that are to be dissolved directly. For example, ground pitchblende and carnotite ore, which may be wrapped in filter paper in the way in which a fusion is wrapped (fig. 7) or sealed in small glass bulbs, which are opened by being crushed against the bottom of the flask by tapping on the glass stem projecting through a second hole in the rubber stopper.[a] To economize time, two of the boiling operations may be carried out simultaneously by the same operator.

METHODS OF RADIUM DETERMINATION APPLICABLE TO VARIOUS SUBSTANCES.

The methods best adapted to determination of the radium in the various products that would usually present themselves for radium analysis in plant control are indicated below.

[a] See Lind, S. C., and Whittemore, C. F., The radium-uranium ratio in carnotites: Tech. Paper 88, Bureau of Mines, 1915, p. 13.

PITCHBLENDE.

High-grade pitchblende is low in silica and readily soluble in hot 1:1 nitric acid. Hence, solution as well as fusion methods are applicable. As the radium-uranium ratio is normal, the radium content may also be calculated from a uranium analysis.

CARNOTITE.

Carnotite is readily soluble in hot 1:1 nitric acid and one of the best methods for its radioactive analysis is solution from a sealed glass tube in which it has been inclosed for a month. Strong ignition (as with Meker burner) of the ore also removes the emanation initially, but second ignition does not do so; therefore, the ignition method is limited to carnotite that has been in a closed container for a month or more. The high silica content of carnotites imparts viscosity to the carbonate fusion, which renders removal of emanation by diffusion difficult. Higher temperature [a] or direct bubbling of air through the fusion doubtless tends to obviate the difficulty, but the authors' experience in general has not been favorable to the use of the fusion method for carnotite. The radium-uranium ratio is normal for large lots of well-sampled ore, and the radium may, hence, be calculated from the uranium content.

CARNOTITE RESIDUES AND TAILINGS.

All the difficulties arising in the analytical treatment of carnotite are manyfold multiplied in the treatment of extracted tailings, with the additional difficulty that as the radium content has already escaped solution, solution methods are not logically applicable. As already stated, the approximate alpha-ray method for solids gives results sufficiently accurate for most purposes (see p. —). In order to apply the emanation method it is necessary, first, to remove the silica from at least a 10-gram sample with hydrofluoric acid before proceeding with the fusion method.

Neither with carnotite nor tailings is it practicable to dissolve the carbonate fusion in acid, as the high silica content soon forms an impervious gel around the surface which prevents further attack.

NITRIC ACID FILTRATE FROM CARNOTITE ORE.

Nitric acid filtrate from carnotite ore may be boiled and sealed directly with or without the addition of barium nitrate, as the original ore contains a large barium excess relative to the radium.

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[a] See Schlundt, H., Trans. Am. Electrochem. Soc., vol. 26, 1914, p. 170.

BARIUM (RADIUM) SULPHATES OR SULPHIDES.

Barium (radium) sulphates or sulphides are fused with carbonate mixture in platinum or porcelain boats, described on page 95, sealed in glass tubes for accumulation, and either dropped into acid or fused directly.

SULPHATE OR CARBONATE FILTRATE.

Sulphate or carbonate filtrate must be handled with all the precautions prescribed for liquids of this character on page 93.

BARIUM (RADIUM) CHLORIDE OR BROMIDE LIQUORS OR CRYSTALS.

Barium (radium) chloride or bromide liquors or crystals may be treated according to the treatment prescribed for a solution containing barium in large excess over radium (p. 92), after suitable dilution for richer fractions from the crystallizing system. The dilution necessary becomes considerable, as much as one to a million in some instances. This dilution is carried out with pipettes and measuring flasks according to the usual methods of volumetric analysis, but involves unusual care in rinsing the vessels used for such large dilution.

The principles already described should suggest a suitable mode of procedure for any other substances that may present themselves for radioactive analysis in connection with the production of radium. But in dealing with any new substance, one should always try several methods for control before final selection.

CONSTRUCTION AND USE OF INTERCHANGEABLE ELECTROSCOPE.

A modified form of the aluminum or gold-leaf electroscope of the C. T. R. Wilson type (Pl. XIV, *B*) has been found entirely satisfactory for all quantitative purposes. The chief modification consisted in making the upper part of the instrument carrying the telescope and leaf system separable from and interchangeable with a large number of gas-tight ionization chambers. This arrangement enables one to carry out a number of emanation determinations in a day without material additional expense, as the emanation chambers can be easily reproduced. The usual type of Wilson electroscope was further modified by building the telescope into an extension front from the leaf chamber, so that the relative positions of electroscope and leaf remain absolutely fixed. Too much stress can not be laid on the importance of this feature. During the year in which an instrument of this type has been in daily use the leaf has not been disturbed, nor has refocusing been necessary; an advantage of the utmost importance

in maintaining the calibration constant. Effort has been made to design a simple instrument that could be constructed, with the exception of the telescope, by any mechanic.

DETAILS OF CONSTRUCTION.

A description of the parts of the electroscope (Pl. XIV, *B*, and fig. 9) and of the emanation chamber follows. The emanation chamber is a gas-tight brass cylinder 4 inches high and 3½ inches in diameter with a volume of about ½ liter. The brass wall of the cylinder is about $\frac{1}{16}$ inch thick, except the bottom plate, which is about ⅜ inch thick. The bottom plate projects ½ inch outside the cylinder and is screwed to a wooden base 6 by 6 inches. This projection also carries a binding post for grounding the instrument. The vertical cylinder projects into the base, into which it is carefully soldered so as to make a gas-tight joint.

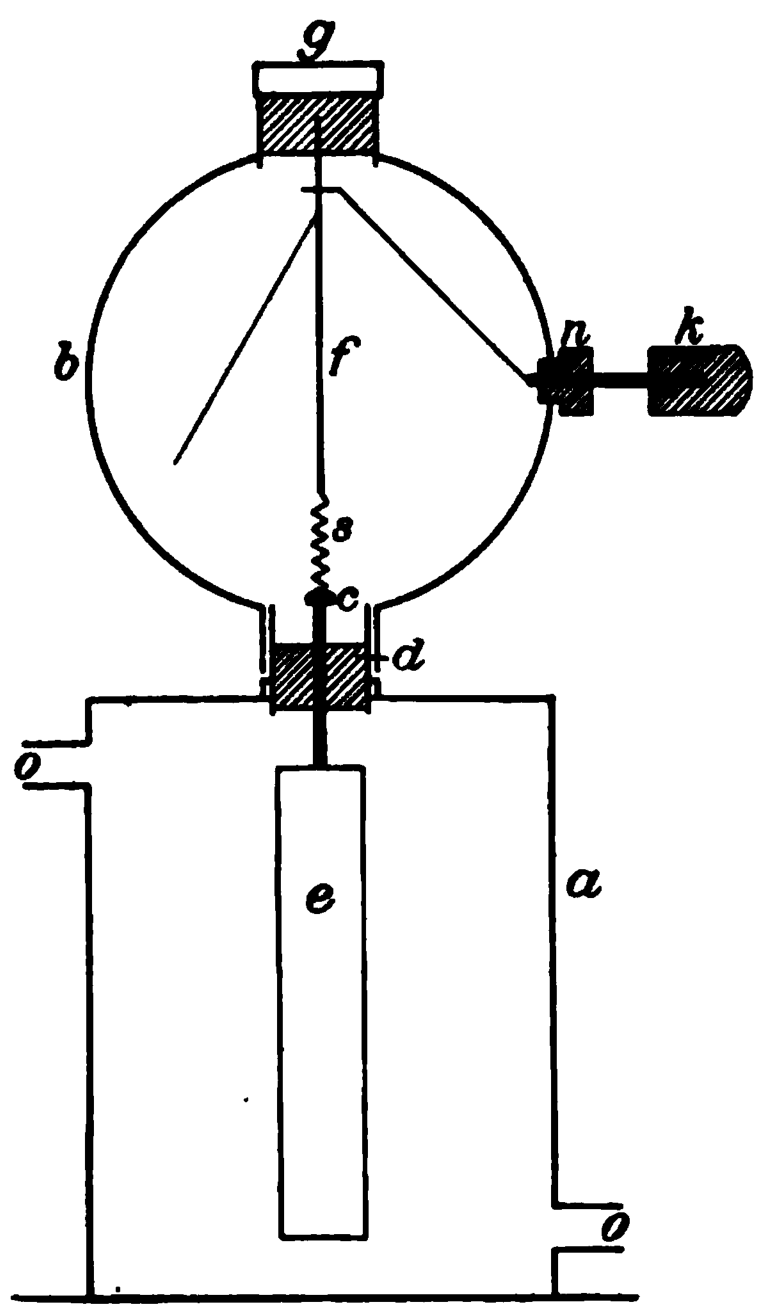

FIGURE 9.—Cross section of electroscope with detachable ionization chamber (one-third size).

On account of the difficulty of obtaining gas-tight brass stopcocks, glass ones have been used and are connected to the plain brass outlet tubes *o* (fig. 9) from the cylinder by means of heavy rubber tubing wired on and with the ends covered with piscein glue. These outlet tubes are ¼ inch in internal diameter and placed ½ inch from the top and bottom on opposite sides of the cylinder.

The electrode *e* is a brass cylinder ½ inch in diameter, projecting downward in the vertical axis of the cylinder to within ½ inch of the bottom and clearing the top by the same distance. The electrode is suspended by a small brass rod ⅛ inch in diameter, which screws into the top of the electrode, passes upward through the insulating material, *d*, and terminates in a small conical cap, *c*, serving to make metallic contact with the leaf system above.

The leaf system *f* is supported from the top of the cylinder where it is held in place by the sealing-wax insulation set in a milled-head cap, *g*, which screws into a vertical collar on the cylinder ¼ inch in height. The cap is hollowed out inside to contain the insulating wax, from which a flat brass rod, *f*, ¼ inch broad, about $\frac{1}{16}$ inch thick,

and $2\frac{1}{4}$ inches long, projects downward, terminating below in a light brass spring, *s*, to make a slight contact with the conical top of the electrode of the ionization chamber. The spring should touch the electrode lightly or it will throw the leaf system out of position.

The aluminum leaf itself, about 2 inches in length, is attached to a small offset at the top of the brass rod by a moisture contact. The whole leaf system may be removed by unscrewing the cap without disturbing the rest of the instrument. If the cap does not screw down tightly into the desired position, a drop of solder may be placed across the joint between the screw head and the collar to prevent accidental displacement of the leaf.

The charging device *k* (fig. 9) consists of a brass rod threaded horizontally through a hard-rubber insulation *n*, in the side of the case. Inside the case the rod slopes upward at an angle of 45° and then extends horizontally, so that contact can be made with the brass rod of the leaf system while being charged, or so that it may be turned and grounded against the wire grating of the outer case.

A collar $\frac{3}{4}$ inch long below the bottom of the cylinder makes a fairly snug contact fit over the collar on top of the discharge chamber and serves as support. A wooden frame is used to hold the upper half of the instrument when it is detached from the base.

One of the best insulating materials for an instrument of this type is high-grade sealing wax, such as "bankers' specie." It has the advantage over sulphur of furnishing both gas-tight connection and good electrical insulation, and is, of course, much less expensive than amber. The sealing-wax insulation is bridged across the bottom of a cylindrical neck $\frac{3}{8}$ inch in internal diameter and 1 inch high above the top of the cylinder. It is desirable to have the minimum layer of wax that will give the necessary strength. A layer $\frac{1}{4}$ inch deep should be ample. The additional height of the neck merely furnishes a friction support for the upper part of the instrument. The electrode and insulation can be removed by unscrewing the whole collar, which is threaded into the upper brass plate of the cylinder $\frac{3}{16}$ inch thick. The collar screws down on a thin lead or rubber washer to insure gas tightness. The removal of the collar and electrode enables one to melt the wax into place with great ease and also to place the electrode in position without disturbing the soldered joint at the bottom of the emanation chamber.

A friction cap fitting snugly down over the neck and the projecting electrode stem protects the insulation from contamination when detached from the upper part of the instrument.

The leaf system and telescope are carried by the upper part of the instrument (Pl. XIV, *B*) and have the advantage over some instruments of being fixed in a perfectly rigid position with respect to each other.

The horizontal cylinder *b* (fig. 9) containing the leaf system is 1¾ inches deep and 3¼ inches in diameter; the ends are closed by sheet mica held in place by steel-wire rings which fit in grooves in the edge of the cylinder in the same way that an automobile tire is held in place. This arrangement has proved most convenient and far preferable to the use of screws. Inside the mica plates and in close contact with them, fine iron-wire gauze serves to conduct off any stray electrical charge. Circular openings in the gauze 1½ inches in diameter furnish a clear field of vision opposite the leaf system.

Opposite the aluminum leaf is a vertical brass plate (not shown in diagram) parallel to the leaf, which may be pushed in so that it almost touches the leaf, and thus protects it from mechanical disturbance during transportation. Whenever the instrument is in use this protector should be withdrawn against the outer case. It may be turned crosswise, if necessary, to remove it as far as possible from the leaf.

Instead of suppporting the telescope on an upright fixed to the same wooden base as the rest of the electroscope, it appeared preferable to fasten it firmly to the case carrying the leaf system. Three arms, such as the one shown in Plate XIV, *B*, carrying a solid brass vertical plate, are firmly screwed onto the case of the leaf system. The telescope fits tightly into a heavy horizontal collar which is screwed into the front plate which is thickened by two small plates to increase the depth of the screw thread. The telescope may be fitted firmly into place and soldered after focusing, or the collar may be split and carry a tightening screw for readjustment.

The telescope used is a Bausch & Lomb type with a 32-mm. objective and a No. 5 eyepiece, carrying a micrometer scale serving to measure the rate of discharge of the leaf. The eyepiece fits firmly into its case, so that its rotation is difficult after the micrometer scale has been set parallel to the leaf.

By means of a charging battery a charge can be maintained for some time on the instrument. Otherwise one can charge with amber or hard rubber, with an ordinary rubber comb for example.

The glass tube (see Pl. XIV, *B*) fixed in the wooden base of the instrument about 1½ inches from the cylinder serves to hold a small sealed tube of radium salt used in controlling the calibration of the instrument. A suitable quantity of radium (about 1 milligram of element, in a sealed tube) furnishes a constant source of penetrating radiation, which may be conveniently employed to control the calibration by measuring the rate of discharge when this tube is placed in the glass tube fixed into the base, and by making comparison with the discharge obtained in the same way at the time of calibration with emanation. Such a measurement can be made in a few minutes and saves a great deal of time in avoiding the repetition of the calibration with emanation, if no marked change is found to have taken place in the rate of discharge.

USE OF ELECTROSCOPE IN EMANATION METHOD.

The principle underlying the use of the emanation electroscope is that, in a given discharge chamber containing at two different times different quantities of radium emanation, the ionization and consequent rate of discharge will be proportional to the quantity of emanation present. If in one case this quantity is known, the unknown quantity can be determined by a direct comparison of the two rates of discharge. The principle seems simple and with the observation of a few essential precautions is really so in application.

Owing to the rapid decomposition of radium emanation into the series of elements, radium A, B, and C, each of which deposits as a solid "active layer" on the walls of the chamber, and contributes materially to the activity, it is necessary to wait three hours after the introduction of the emanation for the active deposit to have reached a maximum. This maximum is maintained with little change between the third and fourth hours, and hence the measurement of the rate of discharge may be made during this period.

It is also to be noted, however, that these active decomposition products of radium emanation carry a positive electrical charge when formed, and hence the position at which they are deposited in the chamber will be somewhat dependent on the electrical field to which they are exposed during deposition, and in turn the ionization and rate of discharge will be influenced. It would seem simple to allow the deposition of the active layer always to take place with no electrical field, but the difficulty here is that as soon as the field is made for measuring the rate of discharge, a shift in the position of newly formed RaA takes place so rapidly that even in the few minutes necessary for measurement, the rate of discharge may change considerably either increasing or decreasing according to whether the new position of RaA is more or less favorable to ionization. To overcome this difficulty one practice is to keep the electroscope charged during the entire three hours of activation, but this procedure may be inconvenient if different instruments are being used on the same charging line, and it has been found more satisfactory in the laboratory to charge for 15 minutes before the measurement. Thus any shift of RaA can be practically completed, and any shift of RaC (through RaB, an α-rayless product) will not have proceeded to any considerable degree.

The measurement itself consists in determining with a stop watch the time elapsing during the passage of the leaf over a certain part of the scale, reading being made always between the same scale divisions. Two or three closely agreeing measurements suffice, but if the deviations are greater than 1 per cent, an average of 10 measurements is taken. The discharge is then reckoned in terms of scale divisions per

second. From the result is subtracted the "natural leak" of the instrument, which is determined before the introduction of emanation. Even with a double contact of wax insulation, the natural leak maintains a low value of about .0003 to 0.005 divisions per second.

Another source of error, which seems especially pronounced in using sealing wax as insulation, is in the so-called "electrical soak" of the insulator, meaning that a certain time is necessary for the insulator to become fully charged. Unless sufficient time is allowed (not less than 15 minutes) for this process to complete itself, the rate of discharge is erratic.

The procedure in the use of the electroscope is then as follows:

1. Set up electroscope as shown in Plate XIV, *B*, and charge for 15 minutes from a battery with just sufficient voltage to hold the leaf on the part of the scale to be used later.
2. Observe the natural leak during 15 or more minutes.
3. Carry out the calibration control by means of penetrating rays if radium is available for this purpose.
4. Detach the top and evacuate the lower chamber to the desired vacuum.
5. Pass the emanation-air mixture through a sulphuric acid drying tube into the evacuated chamber and restore normal pressure.
6. Allow the emanation to stand in the discharge chamber for three hours.
7. Charge for 15 minutes as before.
8. Take three readings if agreements are good, or ten if deviations are greater than 1 per cent.
9. Clean out the emanation chamber by drawing dry, dust-free air through it for some time (over night if convenient).
10. Calculate the discharge and subtract the natural leak, expressing both in divisions per second.
11. Compare the corrected discharge with the calibration of the instrument to determine the quantity of radium under measurement, taking time corrections into consideration.

CALIBRATION OF ELECTROSCOPE.

The calibration of the electroscope is carried out in exactly the same way as in ordinary measurement, except that a known quantity of emanation is introduced. This known quantity may be obtained in two ways, as follows:

1. From a standard solution of some radium salt by passing air through it until its emanation is all transferred into the electroscope. This practice has two disadvantages, the necessity of having and taking care of such a standard solution, and the uncertainty attaching to the quantity of radium emanation removed from it, owing to

the great tendency of radium in such small quantity to be precipitated out in part or to be occluded in the walls of the vessel during prolonged standing. In short, the practice of employing standard radium solutions, though rather general, is not to be recommended, and has been pronounced unsatisfactory at the Radium Institute [a] in Vienna.

2. The preferable practice is to use high-grade analyzed pitchblende, a suitable quantity being dissolved for each standardization, and the quantity of radium being calculated from the uranium analysis. The quantity of radium emanation obtained on dissolving the pitchblende will not correspond exactly to the radium content because a small fraction (2 to 5 per cent) of the gas diffuses from the ore; this fraction, termed the "emanating power," must be determined by sealing a quantity of the ore in a tube for a month or more, and drawing off the emanation into an electroscope by the passage of air. The emanating power thus determined in the standard sample is used as a subtractive correction. Convenient quantities of radium emanation are those that will produce a discharge of the order of 1 to 2 scale divisions per second.

EXAMPLE.

Given a standard pitchblende containing 60 per cent uranium metal and having an emanating power of 3 per cent. If the Ra/U ratio is 3.33×10^{-7}, 1 milligram of pitchblende contains 2×10^{-10}, grams of radium, but as only 97 per cent of this radium can give off emanation, 1 milligram of pitchblende on being dissolved will furnish emanation equivalent to 1.94×10^{-10} grams of radium. For the electroscope herein described use 20 to 40 milligrams of high-grade pitchblende.

Contamination of the discharge chamber may come about through the gradual accumulation of active deposit on the inner walls, which results in the increase of the natural leak of the instrument. For this reason more emanation than is necessary for a measurement is never introduced. The removal of emanation from the chamber should take place at once after the completion of the measurement, and to avoid the introduction of any emanation that may be present in the laboratory air, air is drawn from the outside, being passed through a train of cotton batting to remove dust and through sulphuric acid to remove moisture. Should the discharge chamber become contaminated in spite of all precautions, the chamber is opened and the walls thoroughly washed with dilute (1 to 3) nitric acid, followed by washing with distilled water and drying. This operation is repeated until the natural leak is sufficiently reduced. Contamination of the insulation itself usually necessitates its complete removal.

In measurements of great accuracy it is desirable to calibrate each discharge chamber separately, but by taking greater precaution in

[a] Meyer, Stefen, ———, Sitzungsb. K. Akad. Wiss. Wien, Mathem.-naturw. Klasse, Bd. 122, June, 1913, pp. 1085–1114.

the construction and position of the electrode each chamber can be made to have the same electrical capacity and hence one calibration will serve for all. It seems practicable to reproduce chambers that shall have the same calibration value within 2 per cent.

The convertible electroscope may also be used with other forms of discharge chamber than that used for emanation. For example, it may be used in water analysis by attaching it to a water chamber of the fontactometer type, or may be attached to an open α-ray chamber such as is used for the cursory examination of ores, or to any other desired form of discharge chamber.

ACCESSORIES FOR ELECTROSCOPE.

Accessories for the interchangeble electroscope are listed below.

A wrench for removing the collar carrying the insulation. This is used only in renewing or remelting the insulation. This is done only when the gas leak or the electrical leak has become unduly high. In dry climate, sealing wax dries out rather rapidly, and it is found necessary to remelt the sealing wax about once in one or two months. The operation is simple, but necessitates a new standardization of the chamber.

Two brass dies, used in remelting or replacing the insulation to hold the wax below the electrode.

One brass cap with hole, used in centering the electrode during remelting the insulation.

One hard-rubber cap with brass binding screw head, used in charging one chamber while the regular top is being used on another chamber.

CHAPTER VI. RECOVERIES, LOSSES, AND COST DATA.

RECOVERIES AND LOSSES.

RECOVERY OF URANIUM.

In the extraction process practically all of the uranium in the ore is dissolved in the nitric acid, a little remaining in the insoluble residue because of incomplete washing. An average of 2.3 per cent of the uranium oxide in the ore has remained in the residue, varying in different carload lots from a mere trace to about twice the average. A more thorough washing would remove practically all of this uranium, but would considerably increase the volume of liquid to be handled and evaporated.

As already indicated (p. 50) the loss in the iron-calcium precipitate precipitate has varied, with the amount of sodium carbonate used and the length of time of heating the solution. In the early stages of the work most of the precipitate contained 1½ to 2 per cent U_3O_8 on the dry weight, or 15 to 20 per cent of the oxide in the original ore. This loss was afterwards considerably reduced, mainly by boiling the solution for a longer time and washing the precipitate more thoroughly. The iron-calcium precipitate from some of the carload lots contained as little as 0.48 to 0.50 per cent U_3O_8, representing about 5 per cent of the oxide in the original ore. The U_3O_8 content of the iron-calcium precipitate obtained from the last six carload lots treated averaged 0.7 per cent.

The original plan (see p. 52) called for a double precipitation in order to recover this uranium and the associated radium and vanadium, but it was found that the cost exceeded the value of the products obtained. All efforts were then bent toward raising the yields without reprecipitation.

If, after the precipitation of the sodium uranate, the solution is heated for one hour, practically all of the uranium is precipitated, only a trace appearing with the iron vanadate.

The recovery of the uranium as sodium uranate has, of course, varied with the losses, the extremes being between 75 and 94 per cent. The average on the last 10-carload lots treated has been 84.4 per cent.

RECOVERY OF VANADIUM.

The process described in this bulletin could not be recommended were the recovery of the vanadium in the ore the main object. Hydrochloric acid, under plant conditions, will extract the vanadium

from carnotite more efficiently than will nitric acid, because the reducing action of hydrochloric acid prevents the vanadium from separating as vanadic acid. In this respect even sulphuric acid is better. The chief recommendation for using nitric acid is, therefore, the recovery of radium rather than vanadium. Indeed, it was found in the early operations that any attempt to extract the larger part of the vanadium almost invariably resulted in the precipitation of vanadic acid, retarding filtration and reducing the recovery of radium. Carnotite itself is readily soluble in nitric acid, but the other vanadium minerals present, especially the silicates, are decomposed only after long boiling. A considerable loss of vanadium, therefore, takes place at the start, owing to the insolubility of the vanadium minerals other than carnotite. In order to prevent the separation of vanadic acid, the ore has been heated only as long as the solution remained green, heating being discontinued at once if the solution has showed any tendency to become brown. The main object, therefore, has been to recover radium even at the sacrifice of some of the vanadium. The vanadium content of the ore used has varied from 3.46 to 5.43 per cent V_2O_5, and ore carrying less than 5 per cent presents no difficulties in treatment; however, if this proportion is largely exceeded, the rapidity of filtration is affected, resulting in some loss of radium.

Under the above conditions only about 45 per cent of the vanadium in the ore goes into solution. The iron-calcium precipitate involves a further loss. If sodium carbonate is run into a carnotite acid leach, so that the liquid is acid during most of the time, the vanadium losses will be large even though the carbonate is finally added in excess. Iron vanadates are seemingly formed near the neutral point and after precipitation are redissolved only slowly by the carbonate. In a small experimental plant that used this method to separate the uranium and vanadium from the iron and calcium the iron precipitate obtained carried 15 to 20 per cent V_2O_5. However, if the acid liquor is run into a boiling solution containing an excess of sodium carbonate, as already described, the iron-calcium precipitate obtained need not average more than 2 per cent V_2O_5, and the proportion may be reduced even to 1 to 1.25 per cent under favorable conditions. As explained on page 55, the ruling factor is the length of time the solution is boiled and how much excess carbonate is used. Of course, a point is finally reached where the additional recovery does not pay for the increased expense.

The sodium uranate has carried on an average 8.1 per cent V_2O_5, the limits being 5.2 to 9.4 per cent. If the salt fusion method is used to refine the sodium uranate, practically all of this vanadium is recovered, as the refined sodium uranate on an average carries less than 0.2 per cent V_2O_5. The iron vanadate obtained in the recovery

of this vanadium is of exceedingly high grade, most of it containing about 45 per cent V_2O_5. Seemingly, in the melt the vanadium is completely oxidized to meta-vanadate.

The sodium nitrate filtrate from the iron vanadate may or may not contain vanadium, depending on the method of precipitation. If the precipitation is properly done, the sodium nitrate will carry only a mere trace of vanadium, hardly enough to give a qualitative test.

Lately, on an average 55.5 per cent of the vanadium in the ore remains in the residue, and 13.6 per cent remains in the iron-calcium precipitate, while 8.1 per cent appears in the sodium uranate and 21.4 per cent in the iron vanadate. The total average recovery in vanadium, including that from the sodium uranate is therefore a little less than 30 per cent.

RECOVERY OF RADIUM.

When an element exists in an ore in the proportion of 1 part to 200,000,000, its extraction and recovery present difficulties not ordinarily encountered in metallurgy. A recovery of 60 to 70 per cent or even 50 per cent might, under such conditions, appear to be satisfactory. A much larger recovery than 70 per cent is undoubtedly exceptional. The unusually high recovery of 90 per cent and over of the radium present gives the nitric acid method its real value.

TABULATED DATA.

The table following gives the results of the extraction of the first 21 carloads of carnotite ore received at the plant of the National Radium Institute:

Results of extraction of 21 carload lots of carnotite, from ground ore through first and second sulphates.

[All quantities of radium expressed in terms of milligrams of the element.]

	Ore.					Sulphates.		First sulphates.								Second sulphates.									
Order of treatment.	Car No.	Weight of moist ore.	Proportion of dry U_3O_8.	Proportion of moisture.	Radium in ore.	Batches.	Number of batches.	Weight of first sulphate.	Weight of average batch.	Radium recovered.	Average radium recovery per batch.	Radium per kilogram.	Proportion of radium recovery.	Weight of ore / Weight of sulphate.	Number of re-treatments.	Weight of second sulphate.	Weight of average batch.	Radium recovered.	Average radium per batch.	Radium per kilogram.	Proportion of radium recovery.	Weight of ore / Weight of sulphates.	Number of retreatments.	Radium returned to plant.	Total radium for recovery.
		Pounds.	P. ct.	P. ct.	Mg.			Pounds.	Lbs.	Mg.	Mg.	Mg.	P. ct.			Lbs.	Lbs.	Mg.	Mg.	Mg.	P. ct.			Mg.	Mg.
1	4	43,300	2.51	1.52	137.3	1–19	19	257.0	15.1	116.1	6.1	0.90	84.6	151	6	257.0	13.5	3.3	0.17	0.03	2.4	168	15	....	137.3
2	5	41,400	3.04	1.54	158.8	20–37	18	319.2	17.7	141.3	7.85	0.98	89.0	130	4	324.9	18.0	4.5	0.25	0.03	2.8	127	8	....	158.8
3	3	46,280	2.84	1.80	165.4	38–56	19	330.6	17.4	142.9	7.5	0.95	86.4	140	2	218.1	11.5	7.1	0.37	0.07	4.3	212	7	....	165.4
4	2	33,628	3.93	1.20	167.5	57–70	14	212.6	15.2	90.3	6.45	0.94	53.9	158	5	201.0	14.3	3.7	0.26	0.04	2.2	167	5	....	167.5
5	1	43,250	2.66	1.38	145.2	71–87	17	188.7	11.1	56.2	3.3	0.66	38.7	230	0	146.5	8.6	3.3	0.19	0.05	2.3	296	1	....	145.2
6	P-1	61,[illegible]00	2.27	1.00	177.5	88–110	23	441.2	19.2	142.8	6.2	0.71	80.4	140	4	267.4	10.0	2.3	0.10	0.02	1.3	230	4	....	177.5
7	P-2	80,225	2.69	0.85	274.7	111–142	32	492.9	15.4	244.8	7.65	1.09	89.1	163	3	287.5	9.0	7.3	0.23	0.055	3.0	280	3	....	274.7
8	P-3	74,730	2.79	0.95	264.8	143–171	29	445.4	15.4	242.2	8.35	1.20	89.5	170	1	a88.1	6.8	16.4	0.57	0.41	6.0	362	0	5.72	270.5
9	P-4	93,327	2.75	(1.00)	326.3	172–199B	33	470.4	14.3	297.2	9.0	1.39	85.7	198	0	....	....	....	....	....	....	....	....	20.58	346.9
10	P-5	68,000	2.44	(1.00)	211.0	200A–215B	21	307.6	14.6	206.0	9.8	1.47	95.1	221	0	....	....	....	....	....	....	....	....	5.49	216.5
11	P-6	79,200	2.21	(1.00)	221.8	216A–228B	26	567.1	21.8	200.7	7.7	0.77	90.5	140	11	....	....	....	....	....	....	....	....	0.02	221.8
12	P-7	75,800	2.31	1.00	222.4	229A–240B	23	485.5	21.1	181.8	7.9	0.86	81.7	156	8	198.1	8.6	10.7	0.46	0.12	4.8	388	....	....	222.4
13	P-8	72,100	2.73	1.10	250.0	241B–252B	22	518.6	23.6	240.2	10.9	1.02	94.0	139	8	b168.	16.8	5.5	0.55	0.07	2.15	429	....	5.57	255.6
14	P-9	82,000	2.36	1.30	244.5	253A–264B	24	473.2	19.7	227.6	9.5	1.06	92.4	174	9	97.6	8.1	5.0	0.41	0.11	2.03	840	....	1.86	246.3
15	P-10	77,000	2.24	1.35	218.0	265A–275B	22	382.9	17.4	211.5	9.6	1.22	95.6	201	5	....	....	....	....	....	....	....	....	3.18	221.2
16	P-11	77,000	2.55	1.25	248.5	276A–286B	22	433.2	19.7	226.3	10.3	1.15	89.9	180	3	....	....	....	....	....	....	....	....	3.18	251.7
17	P-12	82,500	2.53	1.23	264.5	287A–298B	24	421.8	17.5	248.9	10.4	1.30	93.0	196	2	....	....	....	....	....	....	....	....	3.10	267.6
18	P-13	84,000	2.81	1.30	299.0	299A–310B	24	470.6	19.6	290.1	12.2	1.35	96.0	180	8	....	....	....	....	....	....	....	....	3.10	302.1
19	P-14	70,000	2.38	1.18	211.3	311A–320B	20	385.3	19.3	197.1	9.85	1.12	92.5	181	....	....	....	....	....	....	....	....	....	1.60	212.9
20	P-15	82,500	2.96	1.13	309.5	321A–332B	24	534.9	22.3	278.6	11.6	1.15	90.0	154	....	....	....	....	....	....	....	....	....	2.80	312.3
21	P-16	80,000	3.02	2.05	303.2	333A–344B	24	482.3	20.1	275.4	11.4	1.25	90.2	166	....	....	....	....	....	....	....	....	....	2.00	305.1
Total...		1,447,940	......	......	4,821.2	....	480	8,651.0	......	4,258.0	......	......	......	......	74	2,254.2	......	60.1	......	......	......	......	43	58.2	4,879.3
Average		68,950	2.66	1.22	229.6	....	....	412.0	19.6	202.8	8.73	1.08	88.3	1.70	3.5	200	11.4	6.3	0.32	0.07	3.0	3.18	4	....	234.4

a 13 batches. b 10 batches. c 12 batches.

DISCUSSION OF TABULATED DATA.

The table shows that with the exception of the ore from two cars, the radium recovered as sulphate has varied from 80.4 to 96 per cent of the radium in the ore with an average of 89.6. If the two cars mentioned are included, the average is 88.3. The average recovery on the last ten carloads treated is 91.5.

In tabulating these results it has been assumed that the radium in the ore is in equilibrium with the uranium, as shown by Lind and Whittemore,[a] and the radium in each carload has been calculated from the uranium oxide content. The radium in the sulphates has always been determined by actual measurement by the emanation method. Results have been accepted only when duplicates gave check results. Although individual sulphate batches were frequently tested, the actual determinations on carload recoveries were made on composites in which the quantity of sulphate used from each batch was proportionate to the total weight of that batch.

During the early part of the work the sulphates were ground and mixed by hand, and a small error in sampling was possible. Later, small ball mills were installed for grinding and thoroughly mixing the sulphates, eliminating the possible error mentioned.

Most of the filtrate from the first sulphate precipitate has carried a little more than 2 per cent of the radium in the ore, as is shown in the column giving the proportion of radium recovery as second sulphates, the filtrate from the second precipitate containing less than 0.1 per cent of the original radium. With four carloads the proportion given was exceeded, the second sulphate from the ore in car P–3 containing 6 per cent of the radium in the ore. The larger part of this was found in three batches and was undoubtedly due to incomplete settling of the first sulphates, a portion of the precipitates being siphoned over and finally appearing with the second sulphate. Indeed, a large part of the average 2 to 2½ per cent loss in the filtrate from the first sulphate is mechanical, not chemical. Efforts are being made to reduce this loss and have already partly succeeded, so well that after the ore in car P–9 had been treated no second sulphates were made.

The ore from cars 1 and 2 gave abnormal results. As regards the ore from car 1, the result can easily be explained, as this ore was part of five cars purchased by the National Radium Institute and had been ground in a ball mill to over 100-mesh fineness instead of being ground to 20 to 30 mesh fineness as is required. Filtration was therefore slow, with a resulting loss of radium. This explanation, however, can not be applied to the ore from car 2, which was readily filtered. Many extraction tests of the ore from this car were made

[a] Lind, S. C., and Whittemore, C. F., The radium-uranium ratio in carnotites: Tech. Paper 88, Bureau of Mines, 1915, pp. 27–28.

in the laboratory, and only 70 to 75 per cent of the radium could be obtained. Although the amount of sulphate in the ore was fairly high, being 0.21 per cent calculated as sulphuric acid, it was no higher than in the ore from car 4 on which the recovery as sulphate was nearly 85 per cent. The delivery during September, 1915, was 407.5 milligrams, making a total to October 1, 1915, of 2,355 milligrams of radium element.

Up to September 1, 1915, 1,947.5 milligrams of radium element had been delivered as high-grade chloride or bromide out of the 4,774 milligrams of radium produced as sulphate. All of this was delivered between February 1 and September 1, 1915, as experimental work on methods of fractionation was not commenced until about December 1, 1914. The grade of the material delivered varied, some containing as much as 87.8 per cent radium. As a rule deliveries were about equally divided between material containing 6 to 15 per cent radium and a higher-grade product containing more than 50 per cent. The lower-grade product is used as a source of emanation for cancer treatment, and the higher-grade product may be used for direct radiation in connection with such work.

A total of 58.2 milligrams of radium from the fractionation process was returned to the plant as barium (radium) chloride. This was low-grade material, the radium content of which was too high to justify its discarding. The large quantity returned in the case of the ore from car P–4 was due to an accumulation from the ore from preceding cars. The total amount of radium discarded in connection with the fractionation of the first 1,646.26 milligrams of element crystallized was 5.13 milligrams, or 0.31 per cent. The radium-barium ratio in this discarded material averaged 25.2 parts per billion.

Except for the discarded liquor mentioned, there are no definite losses that can be indicated quantitatively in refining the radium from the sulphate to the finished product. And yet, there are, of course, small unavoidable losses. The size of these losses will depend largely on the care of the workmen and chemists who do the refining, the personal factor being quite as important as the equipment. In handling and drying the sulphates some small loss through loss of material as dust takes place. In reducing the sulphate with charcoal the evolved gases carry a small amount of sulphate with them. Liquors are occasionally spilt, and "creeping" sometimes takes place in the porcelain and silica vessels in the laboratory. All of these factors can be more or less controlled, and with care the total losses may be kept down to between 1 and 2 per cent. With less careful work they may rise as high as 3 per cent. The loss of one or two crystals of high-grade salt will, of course, have a much greater effect than the careless handling of a large quantity of low-grade material.

The total refining losses at the plant of the National Radium Institute can be only estimated at present. A check on all radium under treatment has been made on more than one occasion, but as nearly 60 determinations were required in each instance the possible total experimental error was much larger than the probable losses. It is believed that the latter have been less than 2 per cent.

RECOVERY OF SODIUM NITRATE.

During the time when crystallization of sodium nitrate from sodium sulphate was necessary, the recovery as usable nitrate was not more than 60 per cent, and sometimes went as low as 50 per cent. With the substitution of nitric for sulphuric acid for a neutralizing agent in the precipitation of the uranium and vanadium, the recovery in nitrate immediately rose to 70 to 75 per cent. More recently the yield has greatly increased, during the months of May, June, and July, 1915, averaging 87.5 per cent, with a minimum of 84.87 per cent and a maximum of 90.04 per cent.

COST OF PRODUCTION.

COST DATA.

During the early stages of the work cost data were kept for all operations, but owing to changes and for other reasons the figures were not subdivided among the several departments in sufficient detail to give the exact data for these departments. Later, this omission was remedied but the departmental figures obtained do not represent an average of the whole operations but rather an average of the results, since the combined plants have been running in connection with the nitric acid plant.

The cost figures include not only expense for labor and materials in connection with the particular operations in question but also the cost of repairs and the actual expense in connection with boiler-room operation, water, electricity, office, laboratory, and superintendence. The figures by departments do not, however, include amortization, insurance, experimental work, cooperation of the Bureau of Mines, and other overhead costs.

The operations have been divided to cover the following items: Leaching, sodium uranate, iron vanadate, sodium nitrate recovery, nitric acid, radium refining, uranium refining, and boiler room.

LEACHING.

Leaching includes operations up to the separation of the barium (radium) sulphate previous to the addition of the acid liquor to an excess of sodium carbonate. The average cost has been $22.62 per ton of ore treated. This item includes the cost of the recovery of

the radium as barium (radium) sulphate. If the recovery of uranium and vanadium was not desired, operations could stop at this point. Such a procedure would vary the conditions in connection with the recovery of sodium nitrate; and as the cost of the nitric acid used is based on a definite recovery of sodium nitrate the above figure of $22.62 would be increased if nitric acid or a large proportion of the sodium nitrate for the manufacture of nitric acid had to be purchased.

SODIUM URANATE.

The figure for sodium uranate includes the cost of all operations in connection with the precipitation, filtering, and drying of the uranium as sodium uranate. The average cost has been 31.49 cents per pound of U_3O_8 in the dry sodium uranate.

IRON VANADATE.

The figure for iron vanadate includes the cost of all operations in connection with the precipitation, filtering, and drying of the iron vanadate. The average cost has been 59.1 cents per pound of V_2O_5 in the dry iron vanadate. This part of the work is therefore conducted at an actual loss, but in combination with the sodium-nitrate recovery, which is based on the extraction of the vanadium, is worked at a profit.

SODIUM NITRATE RECOVERY.

The recovery of sodium nitrate involves the evaporation of the sodium-nitrate solution and the crystallization of the nitrate. The cost per pound of sodium nitrate in the recovered salt has averaged 0.3429 cent.

NITRIC ACID.

The cost of nitric acid has varied with the recovery of sodium nitrate, the market price of nitrate used to make up losses, and the cost of repairs. During the past three months (up to Aug. 1, 1915) it has averaged 2.411 cents per pound of 100 per cent acid, and during the preceding two months 2.050 cents per pound on a basis of 80 per cent recovery of the nitrate.

RADIUM REFINING.

Radium refining includes all operations in the plant in the reduction of the barium (radium) sulphates and the fractionation of the chloride liquors up to the point where deliveries are made to the laboratory. The cost has been $2.44 per milligram of radium element delivered to August 1, 1915. At first it was necessary to get the fractionation systems established both at the plant and at the laboratory. Consequently more radium went into the systems than

came out. The relative cost of refining at the start, based on the quantity of material delivered, was therefore high. More recently the cost has been reduced to $1.02 per milligram. The cost of refining in the laboratory has been $1.03 per milligram of radium element. This figure includes the salaries of the men engaged in the fractionation work and the cost of chemicals, gas, etc., but does not include the cost of analytical work or supervision or other overhead charges.

EXPENDITURES TO AUGUST 1, 1915.

The expenditures to August 1, 1915, exclusive of those for ore and for Bureau of Mines cooperation, are given below. As already stated, the proper distribution of costs on power, water, etc., was not made to departments during the early experimental part of the work, all such costs being placed under "plant operation." Although this was done later on, as already detailed (pp. 113–115), it has been thought better, in giving the figures below, for the sake of uniformity, to still summarize such costs under "plant operation."

Expenditures in plant of National Radium Institute to Aug. 1, 1915.

Construction and equipment:		
Old plant	$21,415.69	
New plant	22,828.65	
Sulphate plant	3,769.31	
Nitric acid plant	4,439.95	
Total charges for construction and equipment		$52,453.60
Operation of plant:		
Old plant (Apr. 1, 1914, to Feb. 1, 1915)	33,250.91	
Combined plants (Feb. 1, 1915, to Aug. 1, 1915)	28,852.90	
Sulphate plant (Oct. 1, 1914, to Aug. 1, 1915)	3,378.01	
Nitric-acid plant (Feb. 2, 1915, to Aug. 1, 1915)	5,973.71	
Chargeable directly to operation		71,455.53
Insurance	1,134.01	
Office expense	897.56	
Laboratory	542.50	
General miscellaneous	2,504.71	
Experimental	919.51	
Chargeable to mines department	2,015.20	
Nitric acid storeroom stock	815.13	
General storeroom stock	3,472.30	
Materials on hand (chemicals, coal, etc.)	3,380.93	
		15,681.85
		139,590.98
Chargeable to mines department	2,015.20	
Nitric acid stock	815.13	
Storeroom stock	3,472.30	
Materials on hand	3,380.93	
		9,683.56
Total		129,907.42

TOTAL COSTS PER GRAM OF RADIUM ELEMENT.

The first ore used in the plant was purchased. This gave an opportunity to test out the process before mining operations were begun. Details of costs of ore mined will be given in a bulletin, being prepared by the Bureau of Mines, on the mining, milling, and concentration of carnotite. These costs cover mining, hauling, freight, grinding, sampling, 70 per cent amortization of equipment, Bureau of Mines cooperation, royalty, and other overhead expenses.

In figuring the total cost of the radium produced, certain overhead expenses, in addition to the cost of the ore and operating costs, that are not included in the operating costs must be charged against the radium. As the plant is in excellent condition and with ordinary repairs, which are taken care of under operating costs, will be perfectly serviceable at least 10 years, an amortization charge for plant and equipment of 20 per cent per annum against each unit from the time it started operation until August 1, 1915, has been made and should be more than ample. The time given to the radium work by the technical men and chemists of the Bureau of Mines has been charged at the full salary rate, a monthly record having been kept of the proportionate amount of his time given by each man. Traveling expenses are also included.

As shown in the table on page 110, 4,258 milligrams of radium element was produced in the form of sulphate from the ore in the first shipments up to and including car P-16, or August 1, 1915. Of this amount, 1,646.56 milligrams of radium element had been delivered, the rest being in the form of untreated sulphate or in process of fractionation. In the table below the cost of refining for delivery these untreated sulphates has been figured at the average refining cost during the last two months considered.

In figuring the cost of the radium, the uranium and vanadium products may be either included or excluded. All the uranium and vanadium has been recovered by the National Radium Institute plant as sodium uranate or uranium oxide and iron vanadate, the whole plant being designed and constructed with this object in view. By subtracting the actual costs connected with the production of the uranium and vanadium compounds from the total cost, a close approximation of the cost of the radium, provided the uranium and vanadium were not recovered, will be obtained, but such a figure will not be exact, as under the changed conditions various factors would enter in, the effects of which can only be estimated. On this basis the average cost of 1 gram of radium element, including the more expensive early treatment, may be determined as follows:

Cost of radium production to Aug. 1, 1915.

Operating costs to Aug. 1:		
Total expenditures	$129,907.42	
Construction and equipment	52,453.60	
		$77,453.82
Cost of ore treated		69,767.99
Amortization of plant and equipment at 20 per cent per annum		9,065.90
Cost of United States Bureau of Mines cooperation (plant department)		13,628.38
Cost of refining 2,611.44 milligrams of radium element (estimated)		5,353.00
		175,269.09
Less costs in connection with the production of the uranium and vanadium compounds		19,947.00
Total cost of 4,258 milligrams of radium element		155,322.09

As already stated, refining losses have almost certainly been less than 2 per cent and probably less than 1 per cent. In order, however, to be on the safe side, an allowance of 3 per cent is made for such losses; 4,258 milligrams less 3 per cent is 4,131 milligrams, which represents the radium finally recovered as high-grade salts. *The average cost of 1 gram of radium element has therefore been $37,599.* It should be remembered that this cost includes the much higher operating costs of the smaller experimental plant and that the first 2 grams of radium extracted cost considerably more per gram than the last 3 grams; also there have been extracted 31,650 pounds of uranium oxide and 11,528 pounds of vanadium oxide. This material has all been contracted for and in part delivered. The returns from its sale will considerably more than cover the cost of its production, and this profit, together with other credits, will ultimately lower by several thousand dollars the cost per gram of radium.

As the price of ore is variable, the question will naturally arise as to the influence of the price of ore on the figures given above. A simple calculation will show at once that the cost of extracting radium, exclusive of the cost of the ore, has been $20,710 per gram. The ore included in the figure of $69,767.99 cited for the cost of ore used in the investigation herein outlined was partly purchased before the war and partly mined by the National Radium Institute at Long Park, Colo. As 723.97 tons was used, the average cost was $96.36 per ton. If the ore had cost $120 per ton, the cost of radium would have been $41,742 per gram, and for every additional amount of $20 per ton above these figures the cost of radium would increase approximately $4,000 per gram.

As it is the hope of the bureau and one of the purposes of this investigation that the miners shall receive a more adequate return for their ore, the figures presented herein will enable anyone interested to determine the approximate cost of radium by the methods outlined under any market price for ore that may prevail.

*

PUBLICATIONS ON MINERAL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines is temporarily available for free distribution. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp.

BULLETIN 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 47. Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the southern Appalachian region, by A. S. Watts. 1913. 170 pp., 16 pls., 12 figs.

BULLETIN 64. The titaniferous iron ores of the United States; their composition and economic value, by J. T. Singewald, jr. 1913. 145 pp., 16 pls., 3 figs.

BULLETIN 71. Fuller's earth, by C. L. Parsons. 1913. 38 pp.

BULLETIN 81. The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs.

BULLETIN 84. Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

TECHNICAL PAPER 3. Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp.

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Bulletin 105

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, Secretary
BUREAU OF MINES
VAN. H. MANNING, Director

BLACK DAMP IN MINES

BY

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AND
G. G. OBERFELL

WASHINGTON
GOVERNMENT PRINTING OFFICE
1916

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First edition. June, 1916.

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BLACK DAMP IN MINES.

By G. A. Burrell, I. W. Robertson, and G. G. Oberfell.

INTRODUCTION.

The Bureau of Mines, in pursuing investigations looking to greater safety in mining, has analyzed samples of the air in many different coal mines in the United States, and has studied the analyses. This report presents the results of one phase of this study and shows how atmospheric air, after entering a coal mine, loses oxygen and gains carbon dioxide with resulting formation of so-called black damp. Also the report discusses the effects of the constituents of black damp on men, on the burning of oil and acetylene lamps, and on the explosibility of methane.

The term "black damp" was and still is widely used to designate accumulations of carbon dioxide, but a more exact definition of black damp, as Haldane, the English physiologist, has pointed out, is an accumulation of carbon dioxide and nitrogen in proportions larger than those found in atmospheric air. The reasons for preferring Haldane's definition are given on succeeding pages.

COMPARISON OF ATMOSPHERIC AIR WITH MINE AIR.

When atmospheric air enters a coal mine, it changes in composition according to (1) the velocity with which it traverses the workings; (2) the amount of coal with which it comes in contact—that is, the extent of the mine workings that it traverses; (3) the gaseous nature of the seam; (4) the tendency of the coal to absorb oxygen; and (5) the temperature and wetness of the mine.

As regards the details mentioned, the governing conditions are as follows: (1) Other things being equal, more carbon dioxide and methane are present and there is a greater deficiency of oxygen at places where the air is still, as at working faces and in old workings. (2) The purity of the air depends on the distance it has traveled, so that in a well-ventilated mine the air is purer at working faces near the shaft than at those that are remote. In the same mine the air will be fresher and purer when the mine is ventilated by a split system than when the air traverses each working face and entry. (3) Some mines vary greatly as regards the generation of methane. Methane is not only

dangerously inflammable, but, if introduced in large proportions at some parts of a mine, it lowers the oxygen content of the atmosphere to such an extent that the atmosphere will not support the combustion of lamps or even will not support life. (4) Coals differ as regards their power of absorbing oxygen. Absorption of oxygen is never accompanied by a molecular proportional increase in the amount of carbon dioxide produced, for the oxygen combines with certain unsaturated hydrocarbon compounds in the coal. (5) The amount of water vapor in air is principally a function of the temperature of the air; hence, if air that is comparatively dry enters a mine, if the temperature of the mine is higher than the outside temperature, and if the mine is comparatively wet, then the proportion of water vapor taken up by the mine air may be greatly in excess of that in the atmospheric air.

In mines other than coal mines the chief factors that decrease the oxygen content of the mine air are the velocity of the air current, the extent of the workings traversed by the air, the amount of oxygen taken from the air, and the amount of carbon dioxide or other suffocating gases added to it by the decay of mine timbers, the burning of lights, the breath of men and of animals, and the oxidation of minerals, and also by the gases given off from the rocks penetrated.

COMPOSITION OF THE ATMOSPHERE.

Pure dry air as analyzed contains, by volume, 20.93 per cent of oxygen, 0.03 per cent of carbon dioxide, and 79.04 per cent of nitrogen. Included in the nitrogen content are the four inactive gases—argon, krypton, neon, and xenon. These, with the exception of argon, which constitutes 0.94 per cent of air, are present in exceedingly small proportions. Water vapor is also present in the air, varying greatly in amount at different times and in different places. The temperature of the air is the most important factor in determining water-vapor content.

Different analysts have found that pure air differs slightly in composition from the figures given above, but these variations, with the exception of carbon dioxide, for which the occasional variation may be one or two hundredths of 1 per cent, are due to unavoidable inaccuracies in the analyses. Pure air has the same composition by volume at sea level or on mountain peaks or at heights that may be reached only with balloons. This constancy of composition is brought about by a gaseous exchange between the plant and the animal life, because animals throw off carbon dioxide, whereas all the higher forms of plant life absorb it. All of the carbon dioxide of the living world comes from the 0.03 per cent of carbon dioxide in the atmosphere.

TRANSFER OF OXYGEN AND CARBON DIOXIDE IN BREATHING.

When a man breathes, air enters the lungs. The oxygen from the air passes through the delicate lining membrane of the air cells of the lungs into the blood, where it forms a loose chemical combination with the hæmoglobin in the red corpuscles. Thus loosely combined the oxygen is conveyed to the heart, and thence through the arteries to the capillaries, where it separates from the hæmoglobin and passes through the capillary walls to the tissues, where it is consumed. Carbon dioxide passes from the tissues in a corresponding manner, forms a loose combination (as bicarbonate and possibly other compounds) in the blood and is conveyed by the veins to the lungs, where the combinations break down, and the carbon dioxide passes from the lining membrane of the lungs into the breath.

EFFECTS ON MAN OF VARIATIONS IN COMPOSITION AND AMOUNT OF AIR.

EFFECT OF CARBON DIOXIDE.

As regards the amount of carbon dioxide present that may give rise to symptoms of poisoning, Haldane and Smith[a] carried out a number of experiments on animals and men and demonstrated that these symptoms did not begin to appear till 3 to 4 per cent of the gas was present, when the breathing became slightly affected. Men can, however, go on working for a considerable time in this atmosphere without feeling serious discomfort, although they will certainly become quickly fatigued, and great exertion will cause panting; but Haldane kept animals for weeks in this atmosphere without causing them much inconvenience. A proportion of 7 to 8 per cent causes more apparent symptoms, and with 10 per cent the distress is great, the headache becomes much more severe, there is marked dyspnœa, throbbing pulse, and flushing of the face, and the gas begins to have a stupefying effect. With 12 to 15 per cent, cerebral symptoms appear, and the patient soon becomes unconscious. Death may take place after exposure for several hours to 25 per cent, but Haldane found that some animals may breathe a much greater percentage, even 50 per cent, without dying.

The concentration of carbon dioxide in the lungs is automatically regulated, so that it is remarkably constant, practically without relation to changes in the rate of breathing, provided respiration is not

[a] Haldane, J. S., and Smith, Lorain, Physiological effects of air vitiated by respiration: Jour. Path. Bact., vol. 1, 1892, p. 174.

forced. This relation was shown by experiments of Haldane and Priestly,[a] the results of which follow:

Results of experiments of Haldane and Priestly to determine relation between rapidity of respiration and concentration of carbon dioxide in lungs.

Subject.	Respirations per minute.	Proportion of carbon dioxide in alveolar air— At end of inspiration.	At end of expiration.	Mean.
		Per cent.	*Per cent.*	*Per cent.*
J. S. Haldane	9	5.59	5.87	5.73
	19	5.56	5.70	5.63
Do	9	5.33	5.47	5.40
	20	5.44	5.60	5.52
J. G. Priestly	10.5	5.95	6.74	6.35
	30	5.98	6.05	6.02

Even during muscular exertion Haldane[a] and Priestly found that when the percentage of carbon dioxide in the inspired air was increased to 4 or 5 per cent the partial pressure of the carbon dioxide in the air in the air cells of the lungs was practically constant.

The effect of breathing air containing carbon dioxide is to stimulate the so-called respiratory center in the brain and increase the depth of breathing. This increase in the depth of breathing is brought about by the smallest increase in the percentage of carbon dioxide in the inspired air, and is sufficient to keep constant the percentage of carbon dioxide in the alveoli, or air cells, of the lungs. This response of the respiratory center of the brain to carbon dioxide is seen during muscular work. The latter increases the production of carbon dioxide in the arterial blood, resulting in an involuntarily increased depth of breathing proportionate to the extra production of carbon dioxide, hence there results the rapid breathing (hyperpnœa) seen during exercise. So small an amount of carbon dioxide in the atmosphere as 1 or 2 per cent does not materially endanger the life and comfort of those who breathe it, but does decrease their efficiency as workmen. A man is forced to breathe a larger amount of air in a given time, and this breathing consumes energy just as the work he is doing consumes energy.

An increase of 50 per cent in the amount of air breathed is brought about by the presence of 1.4 per cent CO_2 in the inspired air, according to Douglas, Haldane, Henderson, and Schneider.[b] They add that panting then becomes excessive even if a man does a great deal less muscular work than he could do in normal air.

[a] Haldane, J. S., and Priestly, J. G., The regulation of the lung ventilation: Jour. Physiol., vol. 32, 1905, p. 225.

[b] Douglas, C. G., Haldane, J. S., Henderson, Y., and Schneider, E. C., Physiological observations made on Pikes Peak, Colo.: Philos. Trans. Roy. Soc. London, ser. B, vol. 203, February, 1913, pp. 220–221.

EFFECT OF DIMINISHED OXYGEN SUPPLY.

Abnormally rapid breathing is also brought about by want of oxygen. In some individuals such breathing is produced when the oxygen percentage of the air is less than 13 (normally about 21) per cent. Rapid breathing is produced much more quickly by excess of carbon dioxide than by a corresponding deficiency of oxygen. The important practical point to remember is that rapid breathing caused by carbon dioxide starts long before danger becomes serious, whereas when it is produced as the result of a deficiency of oxygen it is a grave symptom and points to serious danger.

Bert[a] showed that the abnormal symptoms and dangers associated with low barometric pressure depend, not on the diminished mechanical pressure, but on the diminished partial pressure of the oxygen and consequent imperfect aeration of the arterial blood with oxygen. He found that definite symptoms of want of oxygen began to appear whenever the partial pressure of the oxygen was reduced below a certain limit. Thus at ordinary atmospheric pressure a cat died when the proportion of oxygen was reduced to about 4.5 per cent, or a partial pressure of 4.5 per cent of an atmosphere. At a barometric pressure of 0.5 atmosphere, on the other hand, the animal died when the oxygen was reduced to about 9 per cent, which is again 4.5 per cent of an atmosphere, and at a pressure of 2 atmospheres a reduction of the oxygen percentage to 2.25 per cent was needed. He established the general law that the physiological action of a gas depends on its partial pressure, and further showed that the percentage by volume of oxygen taken up by the blood either in the lungs or outside the body depends on the partial pressure of the oxygen, and is greatly reduced when the lowering of oxygen pressure becomes dangerous.

When men or animals, even though at rest, are subjected to atmospheres deficient in oxygen or at low pressures, marked hyperpnœa is rapidly produced. Marked hyperpnœa caused by oxygen want is only temporary, and is due to the fact that the want aids carbon dioxide in exciting the respiratory center.[b] For a short time the proportion of carbon dioxide already present in the blood and tissues is more than enough to excite the center. When the excess has been eliminated by a temporary hyperpnœa the breathing again becomes quiet, and if the transition to want of oxygen is gradual the temporary marked hyperpnœa is not noticed. Want of oxygen without the aid of carbon dioxide does not excite the respiratory center at all.[c]

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[a] Bert, Paul, La pression barométriqué, 1878.

[b] Haldane, J. S., and Poulton, E. P., Effects of want of oxygen on respiration: Jour. Physiol., vol. 37, 1908, p. 390.

[c] Henderson, Yandell, Apnoea and shock: Am. Jour. Physiol., vol. 25, 1910, pp. 310, 385.

EFFECT OF AIR AT HIGH ALTITUDES.

Men ascend to high altitudes where the air is "thin," and consequently the partial pressure of the oxygen as compared with that at sea level is small. Under the same conditions of temperature and pressure the expansion of different gases is practically the same. Hence the proportion by volume of oxygen, nitrogen, etc., in air remains the same, no matter what the altitude, but the percentage by weight varies as the pressure changes. Air at sea level and at a barometric pressure of 760 mm. contains, by weight, about 23 per cent of oxygen, but at half that pressure, or 380 mm., contains only half of 23 per cent, or 11.5 per cent.

The table following shows the barometric pressures at different heights, also the partial pressure of the oxygen—that is, the percentage of oxygen that would correspond to the same attenuation of air at sea level.

Partial pressures of oxygen corresponding to different barometric pressures.

Altitude.		Inches of mercury.	Barometric pressure, mm. of Hg.	Partial pressure of oxygen compared to pressure at sea level.
Meters.	Feet.			
0	0	30.0	760	21
1,000	3,280	26.0	670	18.5
2,000	6,560	23.4	592	16.3
3,000	9,840	20.6	522	14.4
4,000	12,290	18.2	460	12.7
5,000	16,405	16.0	406	11.2
6,000	19,580	15.4	358	9.9
7,000	22,965	12.5	316	8.7
8,000	26,245	11.0	279	7.7

In explorations of the Himalayas the Duc d'Abruzzi and his companions climbed to a height of 24,580 feet, the barometric pressure being 312 mm., and the partial pressure of the oxygen about 8 per cent.[a] At this point the explorers felt no discomfort during rest and seemingly did not experience any difficulty in performing the work required in climbing.

However, after the explorers had lived for several weeks at a height of 17,000 feet above sea level, the atmosphere did work some harmful effects, revealed only gradually in a slow decrease of appetite and consequent lack of nourishment, but without any disturbance of digestion. Of course, ultimately this insufficient nourishment would have caused a lowering of vitality, loss of flesh, and a certain amount of anæmia. However, the effect was so slow that even at the end of two months the men were able to make long marches without becoming excessively tired.

[a] Filippi, Di Filippo de, Karakoram and western Himalaya, 1909, pp. 361–368.

Many people are affected at a height of 6,000 to 9,000 feet. Everybody suffers from shortness of breath and fatigue at 12,000 feet, and serious symptoms frequently develop at 16,000 feet. At this height the partial pressure of the oxygen corresponds to air at sea level containing about 11.5 per cent oxygen. However, people can live at very high altitudes. In Peru, Bolivia, and northern Chile a large proportion of the population live above 10,000 feet. The elevation of the observatory at El Mista, in the Andes, is about 19,000 feet. It takes time, however, for the body to adapt itself to such an attenuation of the air. A person unaccustomed to it and suddenly plunged into such an atmosphere would experience severe distress.

EFFECT OF BREATHING A DECREASING OXYGEN SUPPLY.

Some instructive experiments on this point, in which one of the authors participated, are described in Technical Paper 122[a] and summarized in Technical Paper 109.[b] The experiments were made under the direction of Yandell Henderson, professor of physiology at Yale University.

A man breathed air in and out of a bag having a capacity of about 70 liters. By means of a can of caustic potash inserted between the man's mouth and the bag the exhaled carbon dioxide was removed. Under these conditions, of course, the oxygen content of the air breathed gradually fell. When it had fallen to about 7 per cent the subject lost consciousness for a few seconds.

The mode of action of the "oxygen want" is instructive. The subject felt no urgent warning symptoms up to the time of collapse. In fact, he wanted to continue, and just before collapse made a show of resistance against discontinuing the experiment. He felt no real distress until some time after the experiment. The next day he was decidedly unwell.

EFFECT OF A MINE ATMOSPHERE LOW IN OXYGEN.

In its insidious action "oxygen want" acts as carbon monoxide frequently does, that is, when the oxygen is slowly decreased. One difficulty in comparing the mode of action of the two lies in the scarcity of experimental data regarding the effects on men of atmospheres containing small percentages of oxygen.

Much light is thrown on the action of the two gases by an accident that occurred in the Lodge Mill Colliery, Huddersfield, England.[c]

[a] Burrell, G. A., and Oberfell, G. G., Effects of atmospheres deficient in oxygen on small animals and on men: Tech. Paper 122, Bureau of Mines, 1915, pp. 7-9.

[b] Burrell, G. A., and Oberfell, G. G., Composition of natural gas used in 25 cities, with a discussion of the properties of natural gas: Tech. Paper 109, Bureau of Mines, 1915, pp. 16-18.

[c] Lloyd, W. D., The use of rescue apparatus at Lodge Mill Colliery, Huddersfield: Coll. Guard., vol. 106, Nov. 7, 1913, p. 957.

Three men were in a disused part of the mine, and two of them, A and B, were overcome by the black damp at 3.30 a. m. A third, C, was overcome at 4 a. m. in an attempt to rescue his comrades. At 1 p. m. of the same day rescuers equipped with breathing apparatus removed C and about an hour later removed A. A little later B was found dead. Both A and C were in a critical condition but still breathing. A died three days later. The air in which the men were overcome contained a high percentage of methane and had a correspondingly low oxygen content. Haldane,[a] in speaking of this disaster, made the following comment:

All the facts recorded indicate that the men were overcome by the insufficiency in the oxygen percentage of the air. There was no reason to suspect the presence of carbon monoxide as there was no gob fire or heating. In addition, the blood of the dead man was black, not red, as it would have been if death had been due to carbon monoxide. It is probable that sufficient fire damp was present to reduce the oxygen to 7 or 8 per cent. The fact that A did not recover was due to exactly the same cause which often prevents men recovering after severe carbon monoxide poisoning. The tissues have been severely damaged by the prolonged exposure to dearth of oxygen, so that, although the oxygen supply is completely restored, recovery is doubtful. In the case of A the post-mortem examination revealed the fact that the heart was dilated. Probably the heart muscles and the other tissues were in a condition of fatty degeneration caused by the want of oxygen. The writer has seen other similar cases of dilatation simulating severe heart disease, and only slowly recovering after prolonged exposure to carbon monoxide poisoning. So far as he is aware, however, this is the only recorded case of death, after partial recovery, from exposure to an atmosphere which was simply deficient in oxygen, apart from the presence of carbon monoxide.

EFFECT OF ATMOSPHERES LOW IN OXYGEN ON CANARIES AND MICE.

Some experiments to determine the effect of atmospheres low in oxygen on mice and canaries, performed by the authors of this bulletin, are described in Technical Paper 122.[b] In conducting the experiments, atmospheres containing various percentages of nitrogen and oxygen were prepared in 10-liter bell jars. For the nitrogen supply a tank containing nitrogen with a small percentage of oxygen was obtained. The atmosphere in the tank analyzed 97.8 per cent nitrogen and 2.2 per cent oxygen.

In the experiments with canaries three birds, designated A, B, and C, were used. Tabulated results of the experiments follow:

[a] Haldane, J. S., Note on paper by W. D. Lloyd: Coll. Guard., vol. 106, Nov. 7, 1913, pp. 957–958.
[b] Burrell, G. A., and Oberfell, G. G., Effect of atmospheres deficient in oxygen on small animals and on men: Tech. Paper 122, 1915, pp. 5–6.

TABLE 1.—*Results of experiments to determine effect on canaries of breathing atmospheres low in oxygen.*

Trial No.	Canary.	Composition of atmosphere during experiment.						Effect on canary.
		At beginning.			At end.			
		Oxygen.	Nitrogen.	Carbon dioxide.	Oxygen.	Nitrogen.	Carbon dioxide.	
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
1	B.....	9.42	90.48	0.10	8.50	90.90	0.60	Immediate distress evinced by rapid breathing, open bill, and tendency to wabble. In 10 minutes the bird was seemingly in normal condition except for slightly increased rate of breathing. It was removed from the atmosphere after 1 hour.
2	C.....	9.25	90.65	.10	....	....	....	Behaved similarly to canary B in trial 1.
3	A.....	7.65	92.25	.10	7.55	92.05	.40	Showed immediate distress by panting and unsteadiness, but did not collapse. It did not evince any more distress than canary B in trial 1. It was left in the atmosphere for 1 hour.
4	C.....	7.83	92.07	.10	....	....	....	Collapsed as soon as it was placed in the atmosphere. It breathed very slowly, with eyes and bill closed, but recovered its normal state in less than a minute after it had been removed to fresh air.
5	A.....	7.10	92.80	.10	....	....	....	Showed distress, but did not collapse in 20 minutes. When canary C was placed in the atmosphere it immediately collapsed.

The next experiments were made with white mice instead of canaries. Tabulated results follow.

TABLE 2.—*Results of experiments to determine effect on white mice of atmospheres deficient in oxygen.*

Trial No.	Composition of atmosphere during experiment.						Effect on mouse.
	At beginning.			At end.			
	Oxygen.	Nitrogen.	Carbon dioxide.	Oxygen.	Nitrogen.	Carbon dioxide.	
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
1	7.07	92.83	0.10	7.75	92.05	0.20	Showed immediate symptoms of distress. At times it appeared normal and again appeared sluggish. Did not collapse. It was left in the atmosphere 1 hour.
2	6.89	93.01	.10	7.12	92.78	.10	Same as on mouse in trial 1.
3	6.70	93.00	.30	....	....	....	Same as on mice in trials 1 and 2.
4	5.90	94.00	.10	....	....	....	Showed immediate distress and collapsed in 6 minutes. One minute after removal from the atmosphere it was on its feet.

GENERAL COMMENTS ON TESTS.

Canaries show some distress in atmospheres containing 9.4 per cent oxygen. The distress is more pronounced in atmospheres containing as low as 7.6 or 7.8 per cent, but even in an atmosphere containing as little as 7.1 per cent of oxygen they may or may not collapse. Mice are slightly more resistant to atmospheres low in oxygen. In an atmosphere containing about 7 per cent of oxygen they show some distress and become more or less sluggish if left in it. One mouse resisted an atmosphere containing as little as 5.9 per cent of oxygen before it collapsed. With that percentage, collapse did not occur until the mouse had been exposed for 6 minutes.

EFFECTS OF OXYGEN DEFICIENCY AS SHOWING RELATIVE VALUE OF BIRDS AND MICE IN EXPLORING MINES.

From the foregoing experiments on birds, mice, and men, it will be seen that oxygen want begins to affect men in about the same time as it does mice and canaries, and the percentages that cause collapse are not much different. Animals differ slightly as to their resistance to collapse in atmospheres low in oxygen, and presumably the same is true of men. The fact is evident that exploring parties in mines can not satisfactorily use birds and mice to detect atmospheres low in oxygen.

Canaries are of chief value in showing by their behavior the presence of carbon monoxide. Afterdamp may contain enough oxygen, about 17 per cent, to support the flame of an oil-fed lamp, so that exploring parties entering it would have no means of knowing that the air is low in oxygen, although the presence of dangerous percentages of carbon monoxide could be detected by the action of the canaries, even if it is not shown by the flame of the safety lamp. When mine-exploring parties, unequipped with breathing apparatus, reach an atmosphere in which oil lamps are extinguished, they are likely to proceed close to or into dangerous atmospheres if they advance farther, even if provided with canaries for detecting vitiated air. The acetylene lamp does not become extinguished until the proportion of oxygen falls to about 13 per cent. In such an atmosphere a party is not in immediate danger, although close at hand there may be atmospheres considerably lower in oxygen, and hence dangerous, into which a person might walk in a very short time. However, any party exploring a mine after a disaster and using canaries would also be equipped with oil safety lamps and with electric lamps, so that acetylene lamps need hardly be considered.

EFFECT OF TEMPERATURE AND MOISTURE ON MAN.

The temperature of the atmosphere is important in that it governs the capacity of air for absorbing moisture. A hot, moist atmosphere prevents the evaporation of perspiration from the body. If, in addition, hot, moist air is stationary, it becomes entangled between the clothing and skin, and becomes warmed to body temperature. Hence, the body can not lose heat to the air, and the skin becomes warmed and bathed in perspiration. Putting a fan in operation whirls the hot, warm air away from the body and allows cooler air to take its place, accounting for the beneficial effect of a fan in a room, even if fresh outside air is not finding access. If the wet-bulb temperature is 99° F. (body temperature), the body will not lose heat, but if the external temperature is 79° or 69° F., perspiration can evaporate. Experience has taught that when the wet-bulb temperature exceeds 75° F. the amount of work that a man can do begins to fall off. Ventilating engineers in this country try to arrange ventilating equipment so that a wet-bulb temperature of 72° F. will not be exceeded. They aim to maintain a relative humidity varying between 30 and 80 per cent. The standard for dry-bulb temperature is about 68° F.

Cadman and Whalley,[a] in an investigation of ventilation in English coal mines, observed that at a wet-bulb temperature of 72° F. heavy clothing was removed and only light clothing worn. At 80° F. hard work was possible, provided the maximum body surface was exposed, the capacity for work being greatly increased, however, if a current of air or a breeze passed over the body. With wet-bulb temperatures of from 80° to 85° F. work was seriously affected, and hard work was almost impossible.

VENTILATION OF BUILDINGS ABOVE GROUND.

In designing the ventilation systems of public buildings, houses, and other places where people meet, the effects of carbon dioxide and oxygen are subordinated to other conditions, such as temperature, humidity, and keeping the air moving. At sea level, in the most ill-ventilated room, the weight of the oxygen in a cubic foot of the air never is as low as in the open air at high altitudes, and it is seldom that the carbon dioxide content rises above 0.10 or 0.20 per cent, although the air in such a room may be unfit to breathe.

The ventilation in rooms is not essentially a matter of correcting diminished oxygen or increased carbon dioxide but of keeping the temperature, the relative moisture, and the movement of the air in a proper state. However, in the ventilation of rooms where people

[a] Cadman, John, and Whalley, E. B., Report of an inquiry into the ventilation of coal mines and the methods of examining for fire damp: Royal Commission on Mines, England, 1909, p. 4.

congregate the carbon dioxide and the oxygen tests may be excellent criterions of the state of ventilation. Haldane and Osborne[a] have investigated the ventilation of a large number of workshops and factories and have recommended that the standard of ventilation be such that the proportion of carbon dioxide does not exceed 0.12 per cent during daylight or 0.20 per cent after dark when oil or gas is used for lighting. The carbon dioxide percentage is specified because its determination is easy and its presence a sign that there is no free ventilation and that the air tends to become humid and warm. However, the vitiation of air in overcrowded rooms may arise from the products of disease, want of cleanliness of the occupants, or the room itself. In fact, air that, judged by the carbon dioxide standard, is sufficiently pure, may be exceedingly impure when judged by the number of microorganisms in it, and vice versa. The test of smell, of feeling of comfort or discomfort, in breathing the air of a room may give results equally at variance with the carbonic acid test. Some people are more influenced by odor, others by an increased quantity of carbon dioxide or moisture or an increased temperature.

Men who habitually work in bad air that is not sufficiently tainted to cause acute symptoms may suffer from slight anæmia, with disinclination for work, rapid breathing on exertion, lassitude, loss of appetite, and other symptoms of indigestion. Return to work in better air, however, soon puts them right.

VENTILATION CONDITIONS IN MINES.

The foregoing discussion of the factors to be considered in ventilation studies has largely to do with conditions aboveground and in public halls, houses, and other places where people congregate. With modifications it is applicable to conditions in coal and in other mines. In meetinghouses aboveground the carbon dioxide in excess of the proportion found in atmospheric air is principally derived from exhaled air, and if present, even in proportions as high as 0.20 per cent, is usually a sign of stagnant air and poor ventilation. It has been found that this proportion of carbon dioxide usually accompanies very bad ventilation in that the air is stagnant and oppressive, with perhaps high temperature and the odor of perspiring bodies.

In coal mines carbon dioxide arises principally from the action of the air on the coal, and 0.2 to 0.3 per cent is frequently found in the cool, swiftly moving air of returns, where 50,000 or more cubic feet of air is passing per minute and the wet-bulb or dry-bulb temperature does not exceed 65° F. The relative humidity may be high, almost 100 per cent in many mines, owing to artificial methods of watering to allay coal dust, but where the temperatures are not high

[a] Haldane, J. S., and Osborne, O. O., Ventilation of factories and workshops: English Govt. Blue Book, 1902.

the high humidity has no bad effect. Hence a proportion of carbon dioxide that would be easily tolerated in a coal mine might be indicative of extremely poor ventilation in the room of a house.

A similar statement applies to oxygen. The oxygen content is scarcely ever normal in a coal mine, owing to the absorption of oxygen by the coal; in fact, a diminution of 1 per cent is not uncommon. Such a diminution in an assembly room would usually be accompanied by intolerable conditions of ventilation. However, the proportion of oxygen can not be diminished too much even in mines. It is the author's experience that as little as 19 per cent of oxygen is rarely found in the moving air of coal mines. However, at some of the working faces of poorly ventilated mines only 17 per cent of oxygen is found.

In mines other than coal mines, carbon dioxide is given off by the burning of lights, by the decay of mine timbers, and, in some mines, by the rocks penetrated. At poorly ventilated working places in some metal mines the carbon dioxide content may be dangerously high or the oxygen content dangerously low, or both conditions may prevail.

As indicated above, an oxygen content as low as 17 per cent is itself without harm, for people live at altitudes where the proportion of oxygen corresponds to this percentage at sea level. The air of many health resorts contains even less oxygen by weight than does the air of Denver, but the air at those resorts is fresh, moving, and invigorating, and usually rather dry. Mine air that contains so small a percentage of oxygen is in no sense comparable to the air of these health resorts. If the air in a part of a certain mine contains only 17 per cent by volume of oxygen not enough fresh air is finding access there; that is, the air is more or less stagnant, perhaps with all the attendant evils discussed previously.

The difference between ventilation conditions in buildings aboveground and in coal mines is in the degree of vitiation of the air as regards low oxygen and high carbon dioxide; aboveground relatively small oxygen and carbon dioxide changes—a few tenths of 1 per cent—usually represent bad air; belowground these small percentages are insignificant. In England, the law requires that the oxygen content shall not fall below 19 per cent or the carbon dioxide content rise above 1.25 per cent in any part of the mine.[a] Mines in this country can easily meet these requirements.

The amount of fresh air per man introduced into coal mines far exceeds that allotted by architects and engineers for ventilating public buildings. As a basis on which to calculate the air necessary for proper ventilation of public buildings, one may consider the

[a] Coal mines act, Great Britain, 1911.

carbon dioxide produced by an adult as 0.6 cubic foot per hour. Hence, in 6,000 cubic feet of air, he will produce per hour a pollution amounting to 1 part of carbon dioxide in 10,000 parts of air. If the excess of carbon dioxide were to be kept down to this figure it would be necessary to supply 6,000 cubic feet of fresh air per hour; if the permissible excess was to be 2 parts in 10,000, half this supply would suffice, and so on. The amount of fresh air allowed by ventilating engineers in this country varies from 1,800 cubic feet per hour per person for school buildings, auditoriums, theaters, factories, etc., to 12,000 cubic feet per hour per person in surgical and contagious hospitals.

In the nongaseous bituminous mines of Pennsylvania the minimum quantity of air per man is 150 cubic feet per minute, or 9,000 cubic feet per hour. In a mine where explosive gas is generated in dangerous quantities the minimum is 12,000 cubic feet per hour per man. These quantities exceed the quantity stipulated in the ventilation of large public halls. Consequently in the moving airways of such mines the air is fresh and wholesome. The temperature is seldom high, and usually bad ventilation exists only at some working faces where the air does not find ready access.

There has been considerable discussion regarding a scheme proposed in England to lower the oxygen content of the air in mines to a point (about 17 per cent) where the risk from explosion of mixtures of methane and air is reduced. Most mining men are opposed to such procedure on physiological grounds. The author's experience has indicated that the oxygen content has to be reduced below this figure to lessen appreciably the liability of gas explosions. (See pages 24–25.)

Some investigators have argued that disastrous coal-dust explosions of recent years can be attributed largely to the better ventilation of mines at present as compared with the ventilation found years ago. To make such a comparison just involves the consideration of too many factors to warrant discussion here. There is truth in the statement, at least as regards mixtures of methane and air, to the extent that greater violence is exerted by an explosive mixture if the latter has a high initial velocity of its own. The authors have seen this demonstrated many times in an explosion gallery of their own construction.[a] But an unbiased consideration of evidence accumulated through long experience must lead one to accept the necessity of plenty of fresh air. Whether for a miner working underground, a man at a health resort, or an assemblage in a large public building, all the fresh air that can reasonably be obtained is needed.

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[a] Details to be published later. The high initial velocities of the gallery tests are, however, probably almost unknown in mines where an explosive mixture of methane and air might exist.

ACTION OF OXYGEN ON COAL.

Many investigations have been made into the mode of absorption of oxygen by coal, which is of much practical importance, for this reaction is mainly responsible for the depletion of oxygen and increase of carbon dioxide in the atmosphere of a coal mine. The detrimental effect is that in some parts of mines atmospheres are produced so low in oxygen that they will not support the combustion of lights or that they impair the health of the miners. By sealing off affected areas, oxygen absorption by coal can be utilized to extinguish mine fires, to prevent spontaneous combustion, or to produce an atmosphere so low in oxygen that explosions of methane and air, or coal dust and air, can not occur.

The exact manner in which coal absorbs oxygen is not clearly understood. A solid substance can hold gases actually dissolved in its interior. With coal, the absorption of oxygen is complicated by the fact that, in addition to this physical action, there is also a very complex chemical action. Part of the oxygen is converted into water, part into carbon dioxide, and part is retained as combined oxygen, producing compounds richer in oxygen than the coal itself. Part of the carbon dioxide is retained by the coal, for coal at ordinary temperatures has an enormous capacity for holding carbon dioxide.

Many investigations have been made into the absorption and liberation of gases by coal, as is indicated by the following selected bibliography:

Selected bibliography on absorption and liberation of gases by coal.

Anderson, W. C. A contribution to the chemistry of coal, with special reference to the coals of the Clyde Basin. Phil. Soc. Glasgow, vol. 29, 1897, pp. 72–96.

Bedson, P. P. The gases inclosed in coal dust. Chem. News, vol. 68, 1893, p. 187.

Boudouard, M. O. Études sur les charbons, produits resultant de leur oxydation. Bull. Soc. chim. France, ser. 4, t. 5, 1909, pp. 377–380.

Chamberlain, R. T. Notes on explosive mine gases and dusts, with special reference to the explosions in the Monongah, Darr, and Naomi coal mines. Bull. 26, Bureau of Mines, p. 32 (reprint of U. S. Geol. Surv. Bull. 383).

Harger, John. The detection of gob fires. Trans. Inst. Min. Eng., vol. 46, 1913–14, p. 370.

Lamplough, F. E. E., and Hill, A. M. The slow combustion of coal and its thermal value. Trans. Inst. Min. Eng., vol. 45, 1913, p. 629.

Parr, S. W., and Wheeler, W. F. Year book of Illinois State Geological Survey. Ill. State Geol. Survey Bull. 8, 1907, p. 168.

Taffanel, J. De l'alteration des poussières de houille exposées a l'air. Original Communications, 8th Int. Cong. Appl. Chem., vol. 10, 1912, p. 277.

Threlfall, Richard. The spontaneous heating of coal, particularly during shipment. Jour. Soc. Chem. Ind., vol. 38, 1909, p. 759.

Trobridge, F. G. Gases inclosed in coal and certain coal dusts. Jour. Soc. Chem. Ind., vol. 25, 1906, p. 1129.

Recent investigations into the action of air on coal are those of Winmill and Graham.[a]

Winmill[b] determined that under his conditions of experiment there was little difference between the rates of oxidation by air, except for mother of coal and slate, between various parts of a seam of coal that he sampled. Hard coal, soft coal, cannel coal, jacks, shale, and mother of coal were tried. He also found that the amount of oxygen absorbed was roughly proportional to the amount of carbonaceous matter present in each substance. One hundred and fifty grams of 200-mesh coal absorbed 169 to 830 c. c. of oxygen at a temperature of 30° C. Winmill states that there are two factors in the oxidation of coal—a rapid first reaction, which is soon over, and a slow reaction, responsible for the major part of the oxygen absorption. He found, as have other investigators, that oxidation of coal, so rapid that the coal heats up, can take place in the absence of pyrites, only a small amount of carbon dioxide being produced. From the hard coal at 30° C. less than 20 c. c. of carbon dioxide was formed during the absorption of 20 c. c. of oxygen. A reduction of the proportion of oxygen to 8.28 per cent (not of the quantity of air) did not slow up the reaction very perceptibly.

With 100-mesh dust the rate of absorption was 72 per cent of that with 200-mesh dust. Increasing the temperature of the experiment from 40° to 60° C. practically doubled the rate of absorption. In another communication Winmill[c] gives the results of some experiments on the rate of absorption of carbon dioxide by coal. The temperature of the experiment was 30° C. The proportion of carbon dioxide absorbed by 100 grams of coal varied from 0.97 c. c., with 0.10 per cent of carbon dioxide in the atmosphere, up to 472 c. c., with an atmosphere containing 100 per cent of carbon dioxide.

INFLUENCE OF MOISTURE ON RATE OF ABSORPTION OF OXYGEN BY COAL.

Fayol[d] maintains that there is little evidence in support of the view that moisture accelerates the firing of coal heaps.

Richters[e] concludes that more oxygen is absorbed by dry than by moist coal.

Mahler[f] found that coal in the presence of oxygen under pressure absorbed a considerably larger quantity when moist than when dry.

[a] Graham, J. I., The absorption of oxygen by coal: Trans. Inst. Min. Eng., vol. 59, Feb., 1915, p. 35.

[b] Winmill, T. F., The absorption of oxygen by coal: Trans. Inst. Min. Eng., vol. 46, 1913-14, pp. 563, 568.

[c] Winmill, T. F., Op. cit., vol. 48, Feb., 1915, p. 514.

[d] Threlfall, Richard, The spontaneous heating of coal, particularly during shipment: Jour. Soc. Chem. Ind., 1909, vol. 28, p. 763.

[e] Threlfall, Richard, Op. cit., p. 759.

[f] Mahler, M. P., Experiments on the oxidation of coal: Coll. Guard., 1913, vol. 106, p. 891.

Graham[a] determined that (1) at temperatures below 50° C. coal dust when moist absorbs oxygen at a rate approximately half as great again as when dry; (2) that coal dust heated for some hours at a temperature of 100° C. in a vacuum during subsequent oxidation absorbs oxygen at the same rate as when not previously heated; (3) that the absorption of oxygen by coal is independent of bacterial activity.

Porter and Ovitz[b] found that 10 kilograms of coal (22 pounds) in a bottle absorbed during the first day after mining nearly half the oxygen from 10 liters of air and gave off little more than one-tenth as much carbon dioxide as would have been formed if all the oxygen had combined with carbon to produce carbon dioxide. Four coals that Porter and Ovitz tested in bottles absorbed in 7, 9, 10, and 17 months 4.362, 2.853, 4.115, and 6.935 volumes of oxygen, as compared with the volumes of the coals. From time to time the gas that accumulated in each bottle was drawn off in order to relieve the pressure and permit the inflow of oxygen.

Porter and Ralston[c] found that with the same coal the rate of oxidation increased from 0.21 c. c. of oxygen absorbed per gram of dry coal at 40° C. to 27.68 c. c. absorbed at 200° C., the duration of each test being 60 minutes. They found that the oxygen content of the air must be reduced below 15 per cent, or to a pressure of about 100 mm. when pure oxygen is used, before the effect on the rate of oxidation becomes serious, and that the presence of as much as 10 per cent of carbon dioxide in the atmosphere has no retarding influence on the rate of oxidation at 200° C.

Winmill, it will be remembered, found that if the percentage of oxygen in air be reduced to 8.20 the reaction did not slow perceptibly, but his results are not in harmony with those of Porter and Ralston. It is certain, of course, that the rate of oxidation must eventually fall decidedly when the oxygen content of the air becomes low, and must be zero when the oxygen content of the atmosphere is zero.

ABSORPTION OF OXYGEN AND PRODUCTION OF CARBON DIOXIDE FROM ATMOSPHERIC AIR BY WOOD.

Many metal mines are heavily timbered and in some mines, especially in dead ends where the air is stagnant, the wood is moist, and much timber is present, the carbon dioxide content of the air is increased and the oxygen content is decreased.

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[a] Graham, J. I., The absorption of oxygen by coal: Trans. Inst. Min. Eng., vol. 59, Feb., 1915, p. 35.

[b] Porter, H. C., and Ovitz, F. K., The escape of gas from coal: Tech. Paper 2, Bureau of Mines, 1911, pp. 6-7.

[c] Porter, H. C., and Ralston, O. C., A study of the oxidation of coal: Tech. Paper 65, Bureau of Mines, 1914, p. 13.

Some results by the authors on the action of air on wood are given in Table 3 following. Dried and seasoned pieces of wood plank were sawed or shaved to produce sawdust or shavings. The material was placed in bottles having a capacity of 2.5 liters and the bottles were securely closed. Through each bottle stopper was placed a glass tube provided with a stopcock to permit drawing out samples of air for analysis.

Results of experiments to determine effect of wood in changing composition of atmospheric air.

Kind of wood.	Weight.	Constituent.	First analysis of residual air, June 27, 1913.	Second analysis of residual air, July 15, 1913.	Third analysis of residual air, July 31, 1913.[a]	Fourth analysis of residual air, Aug. 21, 1913.	Fifth analysis of residual air, Sept. 16, 1913.	Sixth analysis of residual air, Dec. 12, 1913.
	Grams.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Ash sawdust	5	CO_2	0.0	0.15	3.38	11.6	18.9	20.6
		O_2	20.8	20.8	17.50	8.6	1.7	.0
Cypress shavings	76	CO_2	.0	.15	1.98	4.0	6.0	10.0
		O_2	20.9	20.80	18.75	16.7	14.4	10.1
Hemlock sawdust	135.5	CO_2	.20	.20	7.87	16.6	18.0	17.7
		O_2	20.8	20.80	12.00	2.0	.7	.3
Oak sawdust	136	CO_2	.2	.2	8.27	17.0	19.2	20.2
		O_2	20.7	20.8	15.05	3.1	1.2	.4

[a] Water was added to each bottle 15 days after the tests were started and just prior to these analyses.

DISCUSSION OF RESULTS.

The experiments were started on June 6, 1913, and the first analyses of the residual air were made on June 27, 1913. No appreciable change in the oxygen or carbon dioxide content of the residual air over the composition of ordinary atmospheric air was noticed. The same statement held true of results obtained on July 15, 1913, or 39 days after the experiments were started. Water was then added to each bottle in sufficient quantity to perceptibly moisten the sawdust and shavings, and analyses were again made. After 16 days from the time the wood had been moistened, analyses were made of the residual air, and a marked increase in carbon dioxide and a marked decrease in oxygen were found. These changes continued in the sawdust samples until the oxygen had practically all disappeared. The rate of absorption of the oxygen by the fine cypress shavings was slower, probably owing to the fact that the cypress wood was not in as fine a state of division as the other woods tried.

An interesting feature of the results was that the carbon dioxide was only slightly less than the molecular equivalent of the oxygen consumed. This result differs from that obtained when oxygen is absorbed by coal at ordinary temperatures. The amount of carbon dioxide produced by coal is invariably less than the molecular equivalent of the consumed oxygen. In fact, the oxygen may entirely or

almost entirely disappear, and there may be present in the residual air only 2 or 3 per cent of carbon dioxide. Bacterial action apparently does not enter into the phenomenon, whereas as regards the reaction between wood and oxygen, bacterial action is probably almost wholly responsible.

EFFECT OF LOWERING OXYGEN AND RAISING CARBON DIOXIDE CONTENT ON EXPLOSIBILITY OF METHANE-AIR MIXTURES.

In coal mines, especially if the air is not in rapid circulation, the composition of the atmosphere may change quickly. The principal changes that affect the explosibility of mine atmospheres are those caused by the absorption of oxygen by the coal and the oxidation of coal to carbon dioxide. Both of these changes if carried far enough can result in so lowering the oxygen content or raising the carbon dioxide content that an explosion can not take place even if an explosive proportion of methane is present. Therefore, knowledge of the propagation of flame in limit mixtures of methane, carbon dioxide, oxygen, and nitrogen is important.

Haldane[a] was the first to publish results of experiments bearing on this point. The composition of those mixtures containing the smallest percentages of oxygen that he found would completely inflame follows. Haldane presented his results in terms of black damp, fire damp, and air. In the following table his results have been recalculated to show the percentages of carbon dioxide, oxygen, methane, and nitrogen:

Inflammable mixtures containing small percentages of oxygen.

Mixture No.—	CO_2.	O_2.	CH_4.	N_2.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*
1	4.21	11.90	7.46	76.43
2	3.58	13.23	6.85	76.34
3	3.41	13.63	6.46	76.50

Haldane's experiments were made with gas obtained from a coal mine, which was passed into a cylinder, the size of a lamp chimney, from above. Ignition was effected by a small flame from below. Others who have worked on this phase of the subject have been Harger,[b] Leprince-Ringuet,[c] and Burgess and Wheeler,[d] but the

[a] Haldane, J. S., Investigations on the composition, occurrence, and properties of black damp: Trans. Inst. Min. Eng., vol. 8, 1894–95, pp. 549–567.

[b] Harger, John, The prevention of explosions in mines: Trans. Inst. Min. Eng., vol. 43, 1912, pp. 132, 136.

[c] Leprince-Ringuet, F., The inflammability of fire damp and other gases: Coll. Guard., vol. 108, Aug. 14, 1914, p. 376.

[d] Burgess, M. J., and Wheeler, R. V., The propagation of flame in limit mixtures of methane, oxygen, and nitrogen: Jour. Chem. Soc., vols. 105 and 106, November, 1914, pp. 2596–2605.

most exhaustive experiments having to do with the explosibility of methane-air mixtures, in which part of the oxygen of the air was replaced by nitrogen or carbon dioxide or both, have been performed by Clement.[a]

In Clement's first experiments the gases at atmospheric pressure were contained in a Hempel explosion pipette over mercury. The source of ignition, a spark from an induction coil, was applied near the top of the pipette.

Clement found that the limits of inflammability were narrowed as the oxygen was diminished until with 14 per cent of oxygen the low limit was 6.5 per cent of methane and the high limit 6.9 per cent. The inert gas present was nitrogen. When the oxygen was kept constant at 20 per cent and part of the nitrogen replaced by 10 per cent of carbon dioxide the low limit was raised from 5.8 per cent of methane to 6.2 per cent. When the oxygen was again constant at 20 per cent it required the replacement of part of the nitrogen by 62 per cent of carbon dioxide to raise the low limit to 8.8 per cent of methane. Clement found that even when the oxygen was reduced to 17 per cent there was no change in the inflammability of methane-air mixtures from the 5.8 per cent limit observed with 20 per cent of oxygen.

Another set of results was obtained by Clement with a steel tube closed at both ends to hold the mixtures. This tube was provided with stopcocks, an electric-arc igniter, a mixing device, a window for observing the flame, and an opening covered with a paraffin-paper diaphragm through which the pressure was released. The igniter was designed so that an arc could be maintained momentarily or for any desired time. The arc was connected to a 220-volt, direct-current circuit, with suitable resistances in series to give a current of 1.5 amperes, and was placed at the center of the vessel.

With this device the low limit of methane with 19 per cent of oxygen was 5.5 per cent, or 0.3 per cent lower than that found by the first method. With 17 per cent of oxygen the low limit was raised to 5.7 per cent, but even with 13 per cent of oxygen the mixture was explosive with a content of 6.6 per cent as the low limit and 6.8 per cent as the high limit. This second series of experiments again brought out the fact that a large proportion of carbon dioxide was necessary to appreciably affect the limits.

Clement concluded that the action of carbon dioxide in reducing the explosibility of methane-air mixtures can be accounted for by the high specific heat of this gas. He found that carbon dioxide was more effective than nitrogen in reducing explosibility.

The reader should note that atmospheres without enough oxygen (about 17 per cent) to support an oil-fed flame may be explosive if

[a] Clement, J. K., The influence of inert gases on inflammable gaseous mixtures: Tech. Paper 43, Bureau of Mines, 1913, 24 pp.

they contain enough methane. For instance, a miner may travel beyond a place where his flame goes out to a place where the methane is present in explosive proportion, and attempt to relight his lamp there. Its flicker just before it became extinguished would have ignited the mixture, were it explosive. Electric sparks in such an atmosphere would be another source of danger.

The limits of inflammability as determined by two of the authors for mixtures of methane and air in which the methane was simply added to the air were 5.5 per cent for the low limit and 14 per cent for the high limit.[a]

In metal mines, gases that make explosive mixtures with air are seldom found except as the result of a mine fire, but dangerous quantities of methane have been found in some metal-mining operations, as in a tunnel investigated by engineers of the Bureau of Mines.[b]

CONSUMPTION OF OXYGEN AND PRODUCTION OF CARBON DIOXIDE BY MEN AND BY LIGHTS.

A small proportion of the total carbon dioxide in the air of coal mines is that exhaled by men and given off by lights, and the diminution of oxygen is partly due to consumption of oxygen by these agencies. In a sample of air from an air current of a split where 37 men were working, the authors found by analysis 0.32 per cent CO_2, 20.47 per cent O_2, 0.13 per cent CH_4, and 79.08 per cent N_2. The quantity of air passing where the sample was collected was 15,000 cubic feet per minute. If one assumes that each man consumed one-seventh of a cubic foot of oxygen per minute and produced one-seventh of a cubic foot of carbon dioxide[c] during the same time, and that each light consumed about 0.03 cubic foot of oxygen per minute and produced the same quantity of carbon dioxide, there would have been consumed by the men and lights about 6.5 cubic feet of oxygen and they would have produced about 6.5 cubic feet of carbon dioxide. These figures represent only 0.04 per cent of the total oxygen consumed and of the total carbon dioxide produced and are practically negligible compared to the oxygen consumed and the carbon dioxide given off by the action of the oxygen of the air on the coal. Of course, if in this part of the mine, more men had been working, or if the quantity of air passing had been less, appreciable vitiation of air would have resulted from the breathing of the men or the burning of the lights.

[a] Burrell, G. A., and Oberfell, G. G., The limits of inflammability of mixtures of methane and air: Tech. Paper 119, Bureau of Mines, 1915, pp. 25–26.

[b] Brunton, D. W., and Davis, J. A., Safety and efficiency in mine tunneling: Bull. 57, Bureau of Mines, 1914, pp. 29–30.

[c] Actually the oxygen absorbed exceeds in small amount the carbon dioxide discharged; the respiratory quotient, $\frac{CO_2}{O_2}$, being equal to about 0.90.

In most metal mines, lights and the breathing of men and of animals are more important in removing oxygen from and adding carbon dioxide to the mine air, because less oxygen is removed and less carbon dioxide is added in other ways.

FOULING OF MINE AIR BY DECAY OF TIMBER.

In all mines in which timber is used, some oxygen is taken from the air and some carbon dioxide is added to it by the decay of the timber. This source of black damp is much more important in many metal mines than in most coal mines because of the much larger quantity of timber used, especially in heavy ground. Timber decays through the action of various low forms of plant life, chiefly fungi, some of which take oxygen from the mine air, and practically all of which give off carbon dioxide.

SPECIFIC GRAVITY AND COMPOSITION OF BLACK DAMP.

The specific gravity of black damp varies considerably. When methane is present the combined gases may be lighter than air. Great caution should be exercised when any accumulation of black damp lighter than air is found, especially in mines worked with naked lights, as the lesser density is probably due to the presence of methane.

A sample of the following composition was collected by the authors from a cavity in a room of a coal mine:

Results of analysis of sample of mine air of much less density than atmospheric air.

CO_2	0.35
O_2	15.82
CH_4	19.32
N_2	64.51
	100.00

Owing to the large amount of methane present, the specific gravity (air = 1) of this sample was only 0.91.

The analysis of another sample showed an accumulation of methane and a deficiency of oxygen in an inclosed section of an anthracite mine that had been sealed for six days because of a fire in an adjoining section. The fire did not affect the particular area from which the sample was obtained because of a heavy intervening roof fall; consequently the sample represented the gases trapped in a stagnant section unaffected by fire. The results of analysis follow:

Results of analyses of samples of gas in an inclosed area of an anthracite mine.

CO_2	2.6
O_2	3.0
CH_4	53.0
N_2	41.4
	100.0

The specific gravity of this mixture (air = 1) was only 0.77, owing to the large proportion of methane.

Mixtures containing a large amount of carbon dioxide are often found on the floor and in low-lying workings. It happens occasionally that the gas is in a heavy stratum with lighter air above, and these strata are at times so sharply defined that a lighted candle is extinguished at once by lowering it only 1 inch below a certain level.

Equally distinct stratification of gases may be encountered in metal mines. A mixture containing more carbon dioxide, and being consequently heavier than normal air may lie near the floor, or a mixture containing less oxygen and more nitrogen, and being consequently lighter than normal air may accumulate near the roof or in a raise. Either the heavier or the lighter mixture, dangerous to the miner, may be so well stratified that a few inches will make the difference between breathable air and suffocating gases.

Before starting some work near the floor, a miner may hang his lamp well up in the roof, where by burning it indicates that the air near it is breathable; while the air near the floor may be so bad that it would extinguish the lamp immediately. However, except in air directly over a fire area or close to a fire, a large amount of carbon dioxide (over 5 per cent) is unusual in a coal mine. The small proportion of carbon dioxide normally in mine air is indicated by the many analyses of samples presented in this report. Exceptions are samples 5866, 5867, 5785, and 5786 (see p. 40), in which the carbon dioxide percentages were 14.42, 14.36, 19.60, and 19.33, and the specific gravities 1.051, 1.051, 1.071, and 1.069. The samples were collected from sealed mines in which there were or had been fires.

The authors' experience as a result of analyzing gas samples collected from many mines has indicated that if the carbon dioxide results from the action of the oxygen of the air on the coal (including perhaps a small amount released from the pores of the coal), it is not produced in proportions exceeding more than 3 to 5 per cent even after the air has long been in contact with the coal.

The following results of analyses show how small the carbon dioxide content may be in the air of a mine that has been sealed for nine months.

Results of analyses of gas samples from a mine area that had been sealed for nine months.

Sample No.	CO_2.	O_2.	CH_4.	N_2.	Total.	Specific gravity.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	
1	1.50	0.30	5.29	92.91	100.00	0.95
2	1.20	.30	5.37	93.13	100.00	.94

The collectors of the samples wore breathing apparatus. The oxygen had almost entirely disappeared, but only 1.20 and 1.50 per cent of carbon dioxide were present.

EFFECT ON LIGHTS OF AIR LOW IN OXYGEN AND HIGH IN CARBON DIOXIDE.

All mine air is somewhat deficient in oxygen as compared with outside air. This deficiency may amount to 0.1 to 2 per cent or more. As the oxygen in air decreases, the illuminating power of lights grows less; the flame of an ordinary oil-fed lamp wick becomes extinguished in air containing about 17 per cent of oxygen.

Haldane [a] determined the effect on the light of a safety lamp when burned in atmospheres deficient in oxygen. The following tabulation prepared by him is of exceptional interest.

Results of tests to determine diminution of light with diminution of oxygen in surrounding air.

Light given.	Proportion of oxygen in surrounding air.	Degree of light diminution.	Degree of oxygen diminution.
Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
100	20.93	0	0.00
90	20.66	10	0.27
77	20.34	23	0.59
66	19.88	34	1.05
41	19.34	59	1.89
27	18.92	73	2.01
11	18.28	89	2.65
0	18.01	100	2.92

Haldane observed, in general, that every diminution of 0.1 per cent in the oxygen content of the surrounding air caused a diminution of 3.5 per cent of the value of the light in pure air.

The results of some experiments by the authors with atmospheres that would extinguish a flame are shown below. Lighted candles and acetylene and Wolf lamps were placed in a chamber with air, and when the oxygen content had become so low that the flames were extinguished the residual air was analyzed.

Results of experiments with atmospheres that would extinguish flame.

Source of flame extinguished.	Analysis of residual air.		
	CO_2.	O_2.	N_2.
	Per cent.	*Per cent.*	*Per cent.*
Candle	2.95	16.24	80.81
Acetylene lamp	6.30	11.70	82.00
Wolf lamp	3.00	16.30	80.70

[a] Haldane, J. S., The effects of deficiency of oxygen on the light of a safety lamp: Coll. Guard., vol. 106, Oct. 25, 1912, p. 836.

The carbon dioxide present, which was due to combustion, had no appreciable effect in extinguishing the flame, as was shown by certain experiments performed by the authors. In the experiments varying proportions of carbon dioxide were added to the atmosphere before the flame was introduced. The results of analyses made in connection with the experiments follow:

Results of experiments to determine effect of carbon dioxide in extinguishing the flame of a candle.

Experiment No.	Composition of atmosphere at beginning of experiment.		Composition of atmosphere at end of experiment.	
	CO_2.	O_2.	CO_2	O_2.
1............	0.04	21.00	2.95	16.34
2............	3.22	21.13	6.51	16.68
3............	13.52	20.67	16.00	17.39

The results show that the initial presence of a large amount of carbon dioxide had little effect in extinguishing flame. In all of the laboratory experiments the flame probably lasted a trifle longer than it would have in actual mining practice, because in mines a slight gust of air or a quick movement of a lamp would easily put out a diminishing flame.

For the experiments a certain atmosphere was prepared in a 10-liter bell jar, and the flame was allowed to burn in it until the oxygen had been consumed to a point where the flame would no longer burn. Incidentally there was formed by the combustion a certain amount of carbon dioxide, but the amounts formed had no appreciable effect in extinguishing the flame.

Other experiments to determine the amount of carbon dioxide required to appreciably affect flame extinguishment were made, as follows:

Various mixtures of air and carbon dioxide were prepared in a large 100-liter gas holder, and these mixtures were slowly passed into and out of a small bell jar in which was burning a Wolf miner's safety lamp. The different atmospheres were kept as near constant at 17, 18, 19, 20, and 21 per cent of oxygen as possible, and to each of the mixtures carbon dioxide was added (replacing some nitrogen) until enough carbon dioxide was present to extinguish the flame at the particular oxygen percentage used.

The experiments show simply the effect on flame extinguishment of replacing an equivalent amount of nitrogen with carbon dioxide. Carbon dioxide should have a greater extinguishing power than nitrogen, because of its greater molecular specific heat and its greater conductivity of heat. The percentages of carbon dioxide and of oxygen in the different mixtures that extinguished flame are shown in the following tabulation.

Percentages of carbon dioxide and of oxygen in different mixtures of the two that extinguished flame.

CO_2.	O_2.	CO_2.	O_2.
0	16.3	25	18.7
5	16.9	30	19.2
10	17.3	35	19.6
15	17.8	40	20.3
20	18.3	43	21.0

It will be observed that a large percentage of carbon dioxide (replacing the nitrogen) was required to extinguish the flame, 5 per cent of carbon dioxide raising the oxygen content of the extinguishing mixture from 16.3 to 16.9 per cent. This proportion of carbon dioxide in mine air when the oxygen has fallen to 16 or 17 per cent is unusual. It can be said for all practical purposes that the carbon dioxide in mine air has no effect in extinguishing the lamp flames.

Jorrisen [a] has made experiments and compiled data from various sources on flame extinguishment, as shown below:

Results of Jorrisen's experiments with flame-extinguishing atmospheres.

Kind of flame.	Proportions in natural extinguishing atmosphere.		Proportions in artificial extinguishing atmosphere.		
	O_2	N_2	CO_2 added.	O_2	N_2
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Candle	16.4	83.6	14	18.1	81.9
Hydrogen	6.3	93.7	58	8.8	91.2
Carbon monoxide	15.1	84.9	24	16.0	84.0
Methane	17.4	82.6	10	18.9	81.1
Ethylene	13.2	86.8	26	15.5	84.5
Petroleum	16.2	83.8	15	17.9	82.1
Ethyl alcohol	16.6	83.4	14	18.1	81.9
Methylated spirits	17.2	82.8	13	18.3	81.7
Coal gas	11.3	88.7	33	14.1	85.9
Flax	16.0	84.0			

The results of experiments of other investigators, as compiled by Jorrisen, follow:

Results of experiments of various investigators on flame-extinguishing atmospheres.

Nature of flame.	Investigator.	Residual atmosphere.		
		O_2	N_2	CO_2
		Per cent.	*Per cent.*	*Per cent.*
Candle	Clowes	15.70	81.10	3.20
Do	Burrell	16.24	80.81	2.95
Soft candle	Harger	16.90		
Kerosene	Clowes	16.60	80.40	3.00
Do	Harger	16.90		
Petrol spirit	do	16.90		
Gasoline	do	15.50		
Alcohol	Clowes	14.90	80.75	4.35
Methylated spirit	do	15.60	80.25	4.15
Coal gas	do	11.35	83.75	4.90
Natural gas	Burrell	13.90	82.85	3.25
Hydrogen	Clowes	5.50	94.50	
Do	Harger	7.50		
Carbon monoxide	Clowes	13.35	74.40	12.25
Methane	do	15.60	82.10	2.30
Acetylene	Burrell	11.70	82.00	6.30

[a] Jorrisen, W. P. The extinction of flames: Jour. Gas Lighting and Water Supply, vol. 130, Apr. 6, 1915, p. 27.

The agreement of most of the results is fairly satisfactory. The experiments are easy to perform, the principal source of error being insufficient mixing of the atmosphere, with its changing CO_2 and O_2 content, in which the flame is burning, and consequent nonrepresentative sampling.

Jorrisen calls attention to the fact that there is close similarity between the extinguishment of a flame, such as a burning candle or gas jet, in air as the oxygen decreases and the lessening explosibility of gas mixtures as the oxygen diminishes. If the composition of the air surrounding a burning gas jet is near the extinguishing limit, the temperature of the flame is almost instantly lowered so much that the combustible gas supplied can not be ignited, and the flame goes out. Similarly with explosive mixtures of gas and air in a globe, say, when the upper limit of explosion has been exceeded, the oxygen content has become so small that the flame can not extend itself beyond the source of ignition; that is, the mixture will not explode. This is shown in the following table:

Data showing relation between oxygen content of upper explosive limits of mixtures of certain gases and air and oxygen content of similar mixtures that will extinguish flame.

Gas.	Proportion of gas in upper explosive limit.[a]	Proportion of oxygen in gas mixture that extinguishes flame.	Proportion of oxygen at highest explosive limit.
		Per cent.	*Per cent.*
Hydrogen	65	6.50	7.3
Carbon monoxide	74	13.4	5.4
Methane	14.8	17.4	17.8
Gasoline vapor	6.0	15.5	19.6
Natural gas	12.0	13.90	18.4
Acetylene	75	11.70	5.23
Ethylene	22	13.2	16.3

[a] The values in this column were determined by the authors.

The comparison fails badly in the case of carbon monoxide and acetylene.

EFFECT OF ATMOSPHERES LOW IN OXYGEN ON FIRES IN MINES.

The effect of the oxygen content of mine air on a fire in a mine is important [a] and is as follows: When a burning section of a mine has been successfully sealed the atmosphere within changes in composition. It first loses oxygen until the content becomes so low, probably about 17 per cent, that flame can not exist. Eventually the oxygen content becomes so small that the rate of combustion slowly decreases, until ultimately the fire is out and the mine can be safely

[a] See Burrell, G. A., and Seibert, F. M., Gas analysis as an aid in fighting mine fires: Tech. Paper 13, Bureau of Mines, 1912, 16 pp.; Burrell, G. A., and Oberfell, G. G., Explosibility of gases from mine fires: Tech. Paper 134, Bureau of Mines, 1916, 31 pp.; Rice, G. S., Mine fires, a preliminary study: Tech. Paper 24, Bureau of Mines, 1912, p. 38.

opened. Conclusions as to the approximate length of time a sealed mine, or part of a mine, should remain closed after a fire has been brought under control are hard to reach, because conditions differ greatly. When the oxygen in an area has been almost or entirely consumed combustion must necessarily stop, and the question then is how long the embers, or partly burned coal, can retain heat sufficient to permit rekindling on the admission of air. Systematic sampling and analysis of the atmosphere in different parts of a sealed mine will show whether the oxygen diminishes regularly, as it should if stoppings are tightly built, or whether air is leaking into the fire area.

RESULTS OF ANALYSES OF COAL-MINE ATMOSPHERES.

In the course of the investigations reported in this bulletin a large number of analyses of mine atmospheres were made. Methods of analysis are given in Bulletin 42 of the Bureau of Mines.[a] The analyses were classified in series from 1 to 6, and, for convenience of reference, the original classification has been retained in presenting the results.

The following analysis of pure outside air is presented for comparison with the composition of the mine-air samples analyzed.

Analysis of pure air.

	Per cent.
Carbon dioxide (CO_2)	0.03
Oxygen (O_2)	20.93
Nitrogen (N_2)	79.04
	100.00

Included in the nitrogen percentage are the so-called rare gases of the atmosphere—argon, helium, neon, krypton, and zenon. These gases constitute about 1 per cent of the total constituents of air and, like nitrogen, are inert.

In the tabulations that follow, the carbon dioxide, oxygen, methane, and nitrogen, as found by analysis, are given. The results of analyses have been recalculated to show the air, black damp, and methane present and the percentage composition of the black damp. The air content is equal to the nitrogen equivalent of the oxygen, according to the proportions found in pure air plus 0.03 per cent of carbon dioxide. The black damp is equal to the total nitrogen minus the nitrogen equivalent of the oxygen plus the carbon dioxide in the mine air in excess of the amount (0.03 per cent) found in pure air.

[a] Burrell, G. A., and Seibert, F. M., The sampling and examination of mine gases and natural gas: Bull. 42, Bureau of Mines, 1913, 116 pp.

The recalculation of the analysis of a mine-air sample will make clear the foregoing statement. The results of analysis of a mine-air sample follow:

Results of analysis of a sample of air from a coal mine.

	Per cent.
CO_2	0.22
O_2	20.13
CH_4	1.57
N_2	78.08
	100.00

The above results recalculated to show air, black damp, and methane, would give results as follows:

Results of analysis of mine-air sample recalculated to show air, black damp, and methane.

	Per cent.	Per cent.
Air:		
CO_2	0.03	
O_2	20.13	
N_2	76.01	
		96.17
Black damp:		
CO_2	.19	
N_2	2.07	
		2.26
Methane	1.57	
		1.57
		100.00

The composition of the black damp would then be as follows:

CO_2	Per cent.	8.41
N_2	do.	91.59
		100.00

The air in the above results is found as follows:

$$\text{Nitrogen equivalent of the oxygen} = \left(\frac{79.04}{20.93}\right) 20.13 = 76.01$$

$$\text{Air} = 76.01 + 20.13 + 0.03 = 96.17$$

The black damp in the above results is found as follows: The nitrogen in the sample in excess of that found in pure air $= 78.08 - 76.01 = 2.07$ per cent. The carbon dioxide in the sample in excess of that found in pure air $= 0.22$ per cent $- 0.03$ per cent $= 0.19$ per cent. The black damp $= 0.19 + 2.07 = 2.26$ per cent.

RESULTS OF ANALYSES OF SAMPLES IN SERIES 1.

The results of analyses of samples included in series 1 are presented in the following table.

TABLE 4.—*Results of analyses*

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO_2.	O_2.	CH_4.	N_2.
				Per ct.	*Per ct.*	*Per ct.*	*Per ct.*
3734	1	20 feet from face of No. 8 east entry.	Air still; 1-inch cap on safety lamp.	0.35	19.89	1.94	77.82
3735	1	Face of No. 7 south entry..	Trace of methane shown by safety lamp.	.17	20.34	1.05	78.44
3771	2	Crosscut between 2 rooms.	Quantity of air, 4,200 cubic feet per minute; temperature, 61° F., W. B., 60° F., D. B.; relative humidity, 94 per cent.	.08	20.71	.37	78.84
3773	2	.....do.....................	.....do.....................	.11	20.64	.23	79.02
3934	3	(a).........................	(a).........................	7.41	5.76	2.93	83.90
3935	3	(a).........................	(a).........................	7.51	6.20	2.20	84.09
3981	4	No. 1 room, off No. 10 entry, at roof, 20 feet inby.	Air still....................	1.39	16.79	.07	81.75
3982	4	No. 1 room, off No. 10 entry, at floor, 10 feet inby.	.....do.....................	1.17	18.39	.03	80.41
4001	5	Surface crack.............	Crack was over a sealed area in a drift mine that had been on fire a short time previous; fire probably out.	4.82	14.37	.56	80.25
4003	5	Room, 20 feet from a butt, 10 feet from floor.	.....do.....................	7.83	10.69	.89	80.59
4071	6	Main intake for booster fan	....	.35	20.40	.00	79.25
4072	6	Heading of north entry, off 3 east.	....	.48	20.02	.02	79.48
4073	6	16 east, between rooms 56 and 57.	....	.53	20.10	.07	79.30
4074	6	Last crosscut between rooms 19 and 20, No. 16 east entry.	....	1.21	19.39	.07	79.33
4075	6	Main return, at upcast shaft.	....	1.00	19.31	.05	79.64
4008	7	Sixth left entry at face....	9,900 cubic feet of air per minute.	.09	20.44	1.95	77.52
4009	7	Return air course, near exhaust fan.	23,100 cubic feet of air per minute.	.16	20.63	.42	78.79
3339	8	20 feet inside brattice inclosing former fire area.	....	6.02	6.02	[b] .16 3.79	84.01
3340	8	80 feet inside brattice inclosing former fire area.	....	7.74	3.18	[b] .05 3.86	85.17
4138	9	Return air course.........	....	.19	20.61	.00	79.20
4135	9	.....do.....................	....	.21	20.52	.00	79.27
5607	10	Main return...............	27,000 cubic feet of air per minute; temperature 67° F., W. B. and D. B.; relative humidity, 100 per cent.	.31	20.46	.15	79.08
5609	10	Point in entry where air was considerably diluted with return air from fire area.	....	3.07	16.40	2.87	77.66
5602	11	Point in a room where it holed through into a manway.	6,900 cubic feet of air per minute; temperature, 76.5° F., W. B., 78° F., D. B.; relative humidity, 92 per cent.	1.53	18.84	2.16	77.47
5603	11	Manway..................	6,900 cubic feet of air per minute; temperature, 74.5° F., W. B., 77.5° F., D. B.; relative humidity, 87 per cent.	.95	19.89	.66	78.50
5604	11	.....do.....................	7,200 cubic feet of air per minute; temperature, 74.25° F., W. B., 77.25° F., D. B.; relative humidity, 87 per cent.	1.06	19.73	.80	78.41

[a] Sample was collected at the top of a bore hole through which mine air was issuing. The mine was a drift mine that had been sealed for 30 days because of fire. There were, however, many small openings in the outcrop, making impossible complete exclusion of air.

of coal-mine air samples, series 1.

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.		Specific gravity of sample (air=1).
Air.			Black damp.		CH_4	CO_2.	N_2.	
CO_2.	O_2.	N_2.	CO_2.	N_2.				
Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
0.03	19.89	75.11	0.32	2.71	1.94	10.56	89.44	0.99
.03	20.34	76.81	.14	1.63	1.05	7.91	92.09	.99
.03	20.71	78.21	.05	.63	.37	7.35	92.65	.99
.03	20.64	77.94	.08	1.08	.23	6.90	93.10	1.00
.01	5.76	21.75	7.40	62.15	2.93	10.64	89.36	1.00
.01	6.20	23.41	7.50	60.68	2.20	11.00	89.00	1.01
.02	16.79	63.41	1.37	18.34	.07	6.95	93.05	1.00
.03	18.39	69.45	1.14	10.96	.03	9.42	90.58	1.00
.02	14.37	54.27	4.80	25.98	.56	15.59	84.41	1.01
.02	10.69	40.37	7.81	40.22	.89	16.26	83.74	1.02
.03	20.40	77.04	.32	2.21	.00	12.65	87.35	1.00
.03	20.02	75.60	.45	3.88	.02	10.39	89.61	1.00
.03	20.10	75.91	.50	3.39	.07	12.85	87.15	1.00
.03	19.39	73.22	1.18	6.11	.07	16.19	83.81	1.00
.03	19.31	72.92	.97	6.72	.05	12.61	87.39	1.00
.03	20.44	77.19	.06	.33	1.95	15.38	84.62	.99
.03	20.63	77.91	.13	.88	.42	12.87	87.13	1.00
.01	6.02	22.73	6.01	61.28	[b] .16 3.79	8.93	91.07	.99
.00	3.18	12.01	7.74	73.16	[b] .05 3.86	9.57	90.43	1.00
.03	20.61	77.83	.16	1.37	.00	10.46	89.54	1.00
.03	20.52	77.49	.18	1.78	.00	9.18	90.82	1.00
.03	20.46	77.27	.28	1.81	.15	13.40	86.60	1.00
.02	16.40	61.93	3.05	15.73	2.87	16.24	83.76	1.00
.03	18.84	71.15	1.50	6.32	2.16	19.18	80.82	.99
.03	19.89	75.11	.92	3.39	.66	21.35	78.65	1.00
.03	19.73	74.51	1.03	3.90	.80	20.89	79.11	1.00

[b] Laboratory analysis, CO.

TABLE 4.—*Results of analyses*

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO_2.	O_2.	CH_4.	N_2.
				Per ct.	*Per ct.*	*Per ct.*	*Per ct.*
5597	12	Main north return	15,600 cubic feet of air per minute; temperature, 66° F., W. B., and D. B.; relative humidity, 100 per cent.	.34	20.40	.23	79.03
5598	12	....do	....do	.31	20.42	.19	79.08
5599	12	Return of 14 north entry	4,040 cubic feet of air per minute; temperature, 67° F., W. B. and D. B.; relative humidity, 100 per cent.	.12	20.66	.22	79.00
5600	12	....do	....do	.15	20.68	.17	79.00
5578	13	Entry, at face	Temperature, 81.75° F.; W. B., 82.75° F., D. B.; relative humidity, 92 per cent.	.35	20.18	1.25	78.22
5579	13	Seventh south entry, 25 feet from face.	6,000 cubic feet of air per minute.	.29	20.25	1.94	77.52
5580	13	Main south return	8,398 cubic feet of air per minute; temperature, 78.25° F., W. B., 79.25° F., D. B.; relative humidity, 96 per cent.	.33	20.20	1.42	78.05
5581	13	Last crosscut of fifth north entry.	3,780 cubic feet of air per minute; temperature, 81° F., W. B., 83° F., D. B.; relative humidity, 88 per cent.	.35	20.30	1.08	78.27
5583	13	Return at bottom of air shaft.	45,000 cubic feet of air per minute; 76.50° F., W. B., 77.50° F., D. B.; relative humidity, 96 per cent.	.36	20.37	.78	78.49
5586	13	South side return, 100 feet from air shaft.	5,880 cubic feet of air per minute; 78.25° F., W. B., 78.75° F., D. B.; relative humidity, 98 per cent.	.37	20.21	1.56	77.86
5587	13	Main north return, 100 feet from air shaft.	21,250 cubic feet of air per minute; 76.75° F., W. B., 78° F., D. B.; relative humidity, 99 per cent.	.28	20.41	.67	78.64
5499	14	100 feet outby eighth and ninth right.	Sample represented return air from eighth, ninth, tenth, and eleventh right, off north haulage; 2,000 cubic feet of air per minute; 67° F., W. B.; 68° F., D. B.; relative humidity, 95 per cent.	.08	20.44	.95	78.53
5500	14	....do	....do	.12	20.38	1.03	78.47
5501	14	Between eighth and ninth right in last breakthrough.	Air still	.10	20.18	2.04	77.68
5502	14	....do	....do	.16	19.57	4.56	75.71
5503	14	Ninth left, off north haulage, 100 feet from entrance.	Sample of return air from ninth, tenth, eleventh, and twelfth entries, off north haulage; 2,000 cubic feet of air per minute; 67° F., W. B. and D. B.; relative humidity, 100 per cent.	.09	20.39	1.19	78.33
5504	14	....do	....do	.12	20.38	1.03	78.47
5429	15	Third level, close to bulkhead on east side.	Metal mine; 62° F., W. B., 63° F., D. B.; relative humidity, 95 per cent.	.26	20.47	.00	79.27
5430	15	....do	....do	.25	20.44	.00	79.31
5431	15	Seventh level, west, at bulkhead.		.41	20.35	.00	79.24
5432	15	....do		.27	20.53	.00	79.20
5433	15	Upcast shaft, 10 feet below collar.	10,000 cubic feet of air per minute.	.26	20.61	.00	79.13
5434	15	....do	....do	.27	20.60	.00	79.13

of coal-mine air samples, series 1—Continued.

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.		Specific gravity of sample (air=1).
Air.			Black damp.		CH_4.	CO_2.	N_2.	
CO_2.	O_2.	N_2.	CO_2.	N_2.				
Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
0.03	20.40	77.04	0.31	1.99	0.23	13.48	86.52	1.00
.03	20.42	77.11	.28	1.97	.19	12.44	87.56	1.00
.03	20.66	78.02	.09	.98	.22	8.41	91.59	1.00
.03	20.68	78.10	.12	.90	.17	11.76	88.24	1.00
.03	20.18	76.21	.32	2.01	1.25	13.73	86.27	.99
.03	20.25	76.47	.26	1.05	1.94	19.85	80.15	.99
.03	20.20	76.28	.30	1.77	1.42	14.49	85.51	.99
.03	20.30	76.66	.32	1.61	1.08	16.58	83.42	.99
.03	20.37	76.93	.33	1.56	.78	17.46	82.54	.99
.03	20.21	76.32	.34	1.54	1.56	18.09	81.91	.99
.03	20.41	77.08	.25	1.56	.67	13.81	86.19	.99
.03	20.44	77.19	.05	1.34	.95	3.60	96.40	.99
.03	20.38	76.96	.09	1.51	1.03	5.62	94.38	.99
.03	20.18	76.21	.07	1.47	2.04	4.55	95.45	.99
.03	19.57	73.90	.13	1.81	4.56	6.70	93.30	.98
.03	20.39	77.00	.06	1.33	1.19	4.32	95.68	.99
.03	20.38	76.96	.09	1.51	1.03	5.62	94.38	.99
.03	20.47	77.30	.23	1.97	.00	10.45	89.55	1.00
.03	20.44	77.19	.22	2.12	.00	9.40	90.60	1.00
.03	20.35	76.85	.38	2.39	.00	13.72	86.28	1.00
.03	20.53	77.53	.24	1.67	.00	12.57	87.43	1.00
.03	20.61	77.83	.23	1.30	.00	15.03	84.97	1.00
.03	20.60	77.79	.24	1.34	.00	15.19	84.81	1.00

TABLE 4.—*Results of analyses*

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO_2.	O_2.	CH_4.	N_2.
				Per ct.	*Per ct.*	*Per ct.*	*Per ct.*
5075	16	20 feet from face and 20 feet east of room neck, 600 feet west of shaft bottom.	Sample taken in area where pillars were being split; air still.	2.68	13.70	.00	83.62
5076	16	.....do	.....do	2.66	13.74	.00	83.60
5077	16	Main return air course near shaft.	4,700 cubic feet of air per minute.	1.28	17.70	.00	81.02
5078	16	.....do	.....do	1.26	17.77	.00	80.97
5486	17	Manway, 6 feet inby	2,100 cubic feet of air per minute; 76° F., W. B., 79° F., D. B.; relative humidity, 87 per cent.	1.36	18.85	2.08	77.71
5487	17	.....do	.....do	1.41	18.87	2.06	77.66
5490	17	Manway, 200 feet inby	2,100 cubic feet of air per minute; 76° F., W. B., 79° F., D. B.; relative humidity, 87 per cent.	1.37	18.84	2.10	77.69
5491	17	.....do	.....do	1.39	18.86	2.08	77.67
5492	18	North haulage return	54,720 cubic feet of air per minute; 66.5° F., W. B., 66.5° F., D. B., relative humidity, 100 per cent.	.08	20.09	.34	78.89
5493	18	.....do	.....do	.08	20.70	.33	78.89
5465	19	Head of air course	1,900 cubic feet of air per minute.	1.31	18.42	2.09	78.18
5466	19	.....do	.....do	.38	20.30	.53	78.79
5467	19	Main return, inby foot of air shaft.	29,500 cubic feet of air per minute; 75.5° F., W. B., 77.75° F., D. B.; relative humidity, 89 per cent.	1.34	18.92	2.06	77.68
5468	19	.....do	.....do	.42	20.20	.50	78.88
5469	19	15 feet inby F entry	22,500 cubic feet of air per minute; 75.5° F., W. B., 77.75° F., D. B.; relative humidity 90 per cent.	.31	20.44	.48	78.77
5470	19	.....do	.....do	1.31	18.40	1.91	78.38
5472	19	.....do	.....do	1.35	18.66	2.04	77.95
4670	20	No. 1 room, 19th right entry, 4 feet from face and 18 inches from roof.	77° F., W. B. and D. B.; relative humidity 100 per cent.	.10	16.36	5.70	77.84
4671	20	.....do	.....do	.17	16.45	6.38	77.00
4674	20	Main return, 200 feet from pit mouth.	32,277 cubic feet of air per minute; 53.5° F., W. B. and D. B.; relative humidity 100 per cent.	.24	20.44	.14	79.18
4675	20	.....do	.....do	.27	20.45	.13	79.15
4556	21	Return air from 8th and 9th right entries.	66° F., W. B. and D. B.; relative humidity 100 per cent.	.08	20.68	.08	79.16
4557	21	.....do	.....do	.08	20.63	.06	79.23
4558	21	Return air from 8th left entry.	1,000 cubic feet of air per minute; 69° F., W. B. and D. B.; relative humidity 100 per cent.	.09	20.61	.08	79.22
4559	21	.....do	.....do	.09	20.67	.07	79.17
4509	22	Second crosscut, between rooms 1 and 2, 9th right.	Still air; 65° F., W. B., 66° F., D. B.; relative humidity, 95 per cent.	.21	18.94	6.43	74.42
4510	22	.....do	.....do	.21	19.07	6.62	74.10
5903	23	Return of one of two splits, 1,500 feet from pit mouth.	40,100 cubic feet of air per minute; 55° F., W. B., 56° F., D. B.	.05	20.85	.00	79.10
5904	23	.....do	.....do	.08	20.83	.02	79.07
5907	24	Main haulage way, 200 feet from portal.	35,100 cubic feet per minute; 55° F., W. B., 56° F., D. B.; relative humidity, 94 per cent.	.22	20.60	.04	79.14
5908	24	.....do	.....do	.25	20.57	.01	79.17

of coal-mine air samples, series 1—Continued.

Recalculated analysis, showing air, black damp, and methane.						Calculated composition of black damp.		Specific gravity of sample (air=1).
Air.			Black damp.		CH_4.	CO_2.	N_2.	
CO_2.	O_2.	N_2.	CO_2.	N_2.				
Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
0.02	13.70	51.74	02.66	31.88	0.00	7.70	92.30	1.00
.02	13.74	51.89	2.64	31.71	.00	7.69	92.31	1.00
.03	17.70	66.84	1.25	14.18	.00	8.10	91.90	1.00
.03	17.77	67.11	1.23	13.86	.00	8.15	91.85	1.00
.03	18.85	71.19	1.33	6.52	2.08	16.94	83.06	.99
.03	18.87	71.26	1.38	6.40	2.06	17.74	82.26	.99
.03	18.84	71.15	1.34	6.54	2.10	17.01	82.99	.99
.03	18.86	71.22	1.36	6.45	2.08	17.41	82.59	.99
.03	20.69	78.13	.05	.76	.34	6.17	93.83	1.00
.03	20.70	78.17	.05	.72	.33	6.49	93.51	1.00
.03	18.42	69.56	1.28	8.62	2.09	17.37	82.63	.99
.03	20.30	76.66	.35	2.13	.53	13.04	86.96	1.00
.03	18.92	71.45	1.31	6.23	2.06	12.93	87.07	.99
.03	20.20	76.28	.39	2.60	.50	14.11	85.89	1.00
.03	20.44	77.19	.28	1.58	.48	15.05	84.95	1.00
.03	18.40	69.49	1.28	8.89	1.91	12.59	87.41	.99
.03	18.66	70.47	1.32	7.48	2.04	15.00	85.00	.99
.02	16.36	61.78	.08	16.06	5.70	.50	99.50	.97
.02	16.45	62.12	.15	14.88	6.38	1.00	99.00	.97
.03	20.44	77.19	.21	1.99	.14	9.55	90.45	.97
.03	20.45	77.23	.24	1.92	.13	11.11	88.89	1.00
.03	20.68	78.10	.05	1.06	.08	4.50	95.50	1.00
.03	20.63	77.91	.05	1.32	.06	3.65	96.35	1.00
.03	20.61	77.83	.06	1.39	.08	4.14	95.86	1.00
.03	20.67	78.06	.06	1.11	.07	5.13	94.87	1.00
.03	18.94	71.53	.18	2.89	6.43	5.86	94.14	.97
.03	19.07	72.02	.18	2.08	6.62	7.96	92.04	.97
.03	20.85	78.74	.02	.36	.00	5.26	94.74	.99
.03	20.83	78.66	.05	.41	.02	10.87	89.13	.99
.03	20.60	77.79	.19	1.35	.04	12.34	87.66	.997
.03	20.57	77.68	.22	1.49	.01	12.87	87.13	.997

TABLE 4.—*Results of analyses*

Laboratory No.	Source of sample.		Remarks.	Laboratory analysis.			
	Mine No.	Point in mine.		CO_2.	O_2.	CH_4.	N_2.
				Per ct.	*Per ct.*	*Per ct.*	*Per ct.*
5866	25	Air-course entry	Mine sealed when sample was collected because of a previous mine fire.	14.42	3.42	[a] .41	81.42
5867	25	.....do	.....do	14.36	3.63	[a] .44	81.24
5785	25	Fire area in sealed mine, through hole in stopping of outcrop opening.		19.60	.90	[b] 1.95	77.45
5786	25	.....do		19.33	1.18	[b] 1.99	77.40
5818	26	Main return, 75 feet from mine mouth.	45,024 cubic feet of air per minute; 52° F., W. B., 52.5° F., D. B.; relative humidity, 97 per cent.	.07	20.67	.01	79.25
5819	26	.....do	.....do	.09	20.57	.04	79.30
5822	27	Face of right entry, 1,200 feet west of shaft.	3,600 cubic feet of air per minute; 53° F., W. B., 54° F., D. B.; relative humidity, 94 per cent.	.05	20.53	1.70	77.72
5823	27	.....do	.....do	.08	20.50	1.66	77.76
5730	28	Mouth of 7th butt entry	9,800 cubic feet of air per minute; 55° F., W. B. and D. B.; relative humidity, 100 per cent.	.12	20.60	.05	79.23
5731	28	.....do	.....do	.13	20.64	.04	79.19
5732	28	Manway, 300 feet southwest of pit mouth.	40,500 cubic feet of air per minute; 52.5° F., W. B., 53° F., D. B.; relative humidity, 97 per cent.	.07	20.74	.05	79.14
5733	28	.....do	.....do	.09	20.69	.05	79.17
5656	29	Main return, 10 feet from bottom of air shaft.	11,300 cubic feet of air per minute; 66° F., W. B., 67° F., D. B.; relative humidity, 95 per cent.	.71	19.85	.40	79.04
5657	29	.....do	.....do	.72	19.87	.43	78.98
5658	29	8½ east entry, at head of last room through from 9 east entry.	1,200 cubic feet of air per minute; 75° F., W. B., 76° F., D. B.; relative humidity, 96 per cent.	1.80	18.16	1.99	78.05
5659	29	.....do	.....do	2.00	17.78	2.21	78.01

[a] CO, 0.33 per cent. [b] CO, 0.10 per cent.

of coal-mine air samples, series 1—Continued.

Recalculated analysis, showing air, black damp, and methane. Air.			Black damp.			Calculated composition of black damp.		Specific gravity of sample (air=1).
CO_2.	O_2.	N_2.	CO_2.	N_2.	CH_4.	CO_2.	N_2.	
Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
0.00	3.42	12.92	14.42	68.50	*c* 0.33 .41	17.39	82.61	1.051
.01	3.63	13.71	14.35	67.53	*c*.33 .44	17.53	82.47	1.051
.00	.90	3.40	19.60	74.05	*c*.10 1.95	20.93	79.07	1.071
.00	1.18	4.46	19.33	72.94	*c*.10 1.99	20.95	79.05	1.069
.03	20.67	78.06	.04	1.19	.01	3.25	96.75	.996
.03	20.57	77.68	.06	1.62	.04	3.57	96.43	.996
.03	20.53	77.53	.02	.19	1.70	9.52	90.48	.989
.03	20.50	77.42	.05	.34	1.66	12.82	87.18	.989
.03	20.60	77.79	.09	1.44	.05	5.88	94.12	.996
.03	20.64	77.94	.10	1.25	.04	7.41	92.59	.994
.03	20.74	78.32	.04	.82	.05	4.65	95.35	.996
.03	20.69	78.13	.06	1.04	.05	5.45	94.55	.996
.03	19.85	74.96	.68	4.08	.40	14.29	85.71	.997
.03	19.87	75.04	.69	3.94	.43	14.90	85.10	.997
.03	18.16	68.58	1.77	9.47	1.99	15.75	84.25	.994
.03	17.78	67.14	1.97	10.87	2.21	15.34	84.66	.994

c Laboratory analysis, CO.

COMMENT ON ANALYSES OF SAMPLES IN SERIES 1.

The results of analysis of 111 samples of mine air from 29 coal mines are shown in the preceding table. The percentage of black damp in the samples varies from 0.14 to 82.92. The average percentage of carbon dioxide in the black damp is 11.5 per cent, and the average percentage of nitrogen is 89.5 per cent.

The principal factors that probably affect the formation of black damp in coal mines are (1) time of contact between the air and coal; (2) rate of reaction of a particular coal with oxygen, with subsequent formation of some carbon dioxide; (3) temperature of the coal and the air, and (4) presence of moisture. Of these the two first are undoubtedly the most important; hence it is to be expected that different percentages of black damp will be found at different places in the same mine. In a return airway, where the velocity of the air current is high, and the air has been in contact with the coal only a short time, small percentages of black damp are found; however, in the same mine, at a working face, where the air travels slowly, or in an abandoned part of a mine, where the air is stagnant, the percentages of black damp may be high, conditions clearly indicated by many of the results of analyses in the foregoing tables.

If all coals upon reaction with air produced carbon dioxide in the same ratio to the oyxgen consumed, then the percentage composition of black damp should be always the same in all mines. Laboratory experiments show, however, that there is no such constancy, as some coals produce more carbon dioxide in relation to the oxygen absorbed than do others; hence black damp formed by the action of air on different kinds of coal must vary in composition.

A question of interest arises, Should the ratio between the carbon dioxide produced and the oxygen consumed be always the same for each coal? If one takes a sample of fresh coal and treats different portions of it with oxygen under the same conditions, practically identical results will be obtained. If, however, one portion is allowed to weather a long time, or if more moisture is present in one than in another, the results are not so concordant. Probably, also, other factors must be taken into consideration. The analyses given in the preceding table show a fair degree of concordance; that is, samples of black damp from different parts of the same mine do not vary as regards percentage composition of the black damp nearly so much as do samples from different mines, although there is lack of strict concordance which shows that even in the same mine different samples differ in composition.

TEMPERATURE AND HUMIDITY.

Wet and dry bulb temperature readings were taken in 22 of the 29 mines represented in the first series of results. In 6 mines of the 22 the temperature was higher than it should be under the best ventilating conditions. In mine 11 the wet-bulb temperature at one place varied from 74.25° to 78.5° F. However, this place was used only for traveling. The wet-bulb temperature in mine 13, both at the face and in the moving air current, was too high. The proportion of methane was also higher than it should be, ranging from 0.67 per cent in the return 100 feet from the air shaft to 1.94 per cent 25 feet from the seventh south entry. In other words, the introduction of more fresh air into this mine would not only have cooled it, but would have diluted the methane present.

Mine 16 was poorly ventilated. In the return air course where only 4,700 cubic feet of air was passing (samples 5077 and 5078) the carbon dioxide averaged 1.27 per cent and the oxygen 17.74 per cent. More air would certainly have been desirable.

In mine 18 there were high temperatures combined with high carbon dioxide, low oxygen, and high methane content in the air of the manway.

In mine 20, the oxygen content of the air was low, the carbon dioxide and methane contents high, and the temperature high.

RESULTS OF ANALYSES OF SAMPLES IN SERIES 2 TO 6.

In Table 5 following are presented the results of analyses representing mine air collected in different mines in this country. The results have been divided into five series following series 1 in the foregoing table. The samples in series 2 and 3 were collected in mines in West Virginia. The samples in series 4 were collected in an anthracite mine in eastern Pennsylvania, and the samples in series 5 and 6 were collected in two different mines in Indiana. In each mine the sampler started at the intake, and traveled in the direction of the air, collecting samples at certain intervals, so as to show the progressive change in the mine air as it traversed the mine and came in contact with more and more coal.

The table shows the laboratory number of each sample; the place where the sample was collected; the composition of the sample as received; the recalculated analysis showing the atmospheric air, black damp, and methane in the sample; and the composition of the black damp. Certain observations made in connection with the different samples are also given.

TABLE 5.—*Results of analyses of*

SAMPLES IN SERIES 2, FROM

Bottle No.	Sample No.	Distance of point of sampling from intake shaft.	Point of sampling.	Quantity of air per minute.	Temperature at point of sampling.		Relative humidity.
					Wet bulb.	Dry bulb.	
		Feet.		*Cubic feet.*	°*F.*	°*F.*	*Per ct.*
1	6358	2,475	Main air course of west heading	62,040	54	54	100
2	6357	3,000	Air course, off Green heading	16,884	54	54	100
3	6355	3,400	Air course, off Green heading, between rooms 5 and 6	20,216	54	55	94
4	6356	3,800	Air course, off Green heading, between rooms 14 and 15	13,017	53	54	94
5	6354	4,200	Air course, off Green heading, opposite room 24	9,928	53	55	88
6	6361	4,600	Air course, off Green heading, opposite room 33	4,260	54	56	88
7	6353	5,000	Air course, off Green heading, opposite room 42	Air still.	54	56	88
8	6360	5,575	Ritter heading, opposite room 48	5,266	56	57	94
9	6362	6,075	Ritter heading, right side, opposite room 1	Air still.	55	56	94
10	6352	6,250	Ritter heading, room 55, off left side, at face	Air still.	56	58	89
11	6359	6,275	Ritter heading, at face of room 2, off right side	Air still.	55	56	94
12	6350	6,575	Ritter heading, 30 feet from upcast air shaft	5,280	55	55	100

SAMPLES IN SERIES 3, FROM

Bottle No.	Sample No.	Distance of point of sampling from intake shaft.	Point of sampling.	Quantity of air per minute.	Wet bulb.	Dry bulb.	Relative humidity.
1	6372	1,012	12 left air course	Air still.	57	58	94
2	6365	1,512	Face of air course of X entry, off tenth left heading	Air still.	58	59	94
3	6366	1,812	Face of room 3, off tenth right heading	Air still.	58	60	89
4	6369	2,638	Opposite first cross-cut beyond twelfth left heading	Air still.	58	59	94
5	6373	3,438	Between rooms 2 and 3, off twelfth left heading	Air still.	57	58	94
6	6367	3,650	Room 3, off twelfth left heading	Air still.	57	58	94
7	6368	3,738	Room 7, off twelfth left heading	Air still.	58	60	89
8	6374	4,250	Between first and second crosscuts	Air still.	57	58	94
9	6375	5,025	Tenth left entry, opposite fifth crosscut	1,750	57	58	94
10	6370	5,812	Main air course at intersection with ninth right heading	29,000	57	58	94
11	6377	6,612	Main air course, between third and fourth crosscuts, beyond eighth right heading	24,450	56	57	94
12	6378	7,425	Main air course, 40 feet beyond seventh left heading	41,800	57	58	94
13	6376	8,075	Main air course, 60 feet in by fan	61,900	50	54	76

SAMPLES IN SERIES 4, FROM AN ANTHRACITE

Bottle No.	Sample No.	Distance of point of sampling from intake shaft.	Point of sampling.	Quantity of air per minute.	Wet bulb.	Dry bulb.	Relative humidity.
1	6293	200	200 feet from slope	6,400	61	63.5	87
2	6290	470	470 feet from foot of slope	8,800	61	64.5	82
3	6296	682	682 feet from foot of slope, at face	Air still.	61	63.5	87
4	6297	802	802 feet from foot of slope	12,300	61	63.5	87
5	6298	1,527	1,527 feet from foot of slope	10,500	62	64.5	87
6	6301	2,242	2,242 feet from foot of slope	9,700	62	64.5	87
7	6289	3,207	3,207 feet from foot of slope	8,900	64.5	66	87
8	6302	4,320	3,800 feet from foot of slope	6,310	62	64.5	87
9	6292	4,840	4,320 feet from foot of slope	4,360	63	66	85
10	6300	5,110	5,110 feet from foot of slope	Air still.	63	66	85
11	6299	5,915	5,915 feet from foot of slope	7,170	63	66	85
12	6294	6,575	6,575 feet from foot of slope	15,300	63	66	85

mine-air samples, series 2 to 6.

A MINE IN WEST VIRGINIA.

Composition of sample as received.				Recalculated analysis to show air, black damp, and methane.						Composition of black damp.		Specific gravity (air=1).
				Air.			Black damp.		Methane.			
CO_2.	O_2.	CH_4.	N_2.	CO_2.	O_2.	N_2.	CO_2.	N_2.	CH_4.	CO_2.	N_2.	
P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
0.05	20.83	0.03	79.09	0.03	20.83	78.66	0.02	0.43	0.03	4.4	95.6	1.00
.06	20.80	.03	79.11	.03	20.80	78.54	.03	.57	.03	5.0	95.0	1.00
.06	20.85	.04	79.05	.03	20.85	78.73	.03	.32	.04	8.6	91.4	1.00
.07	20.86	.02	79.05	.03	20.86	78.77	.04	.28	.02	12.5	87.5	1.00
.04	20.90	.02	79.04	.03	20.90	78.92	.01	.12	.02	7.7	92.3	1.00
.10	20.76	.00	79.14	.03	20.76	78.39	.07	.75	.00	8.5	91.5	1.00
.11	20.75	.02	79.12	.03	20.75	78.36	.08	.75	.03	9.6	90.4	1.00
.11	20.71	.03	79.15	.03	20.71	78.20	.08	.95	.03	7.8	92.2	1.00
.09	20.81	.00	79.10	.03	20.81	78.58	.06	.52	.00	10.3	89.7	1.00
.29	20.38	.02	79.31	.03	20.38	76.96	.26	2.35	.02	10.0	90.0	1.00
.10	20.77	.05	79.08	.03	20.77	78.43	.07	.65	.05	9.7	90.3	1.00
.15	20.57	.04	79.24	.03	20.57	77.85	.12	1.56	.04	7.1	92.9	1.00

A MINE IN WEST VIRGINIA.

0.47	19.72	0.00	79.81	0.03	19.72	74.47	.44	5.31	0.03	7.7	92.3	....
.52	19.64	.02	79.82	.03	19.64	74.17	.49	5.65	.02	8.0	92.0	....
.42	19.81	.02	79.75	.03	19.81	74.81	.39	4.94	.02	7.3	92.7	....
.46	19.98	.00	79.56	.03	19.98	75.45	.43	4.11	.00	9.5	90.5	....
.43	19.82	.00	74.85	.03	19.82	74.85	.40	4.89	.03	7.6	92.4	....
.45	19.66	.02	79.87	.03	19.66	74.02	.42	5.85	.02	6.7	93.3	....
.67	19.54	.01	79.78	.03	19.54	73.79	.64	5.99	.01	9.7	90.3	....
.46	19.63	.08	79.83	.03	19.63	74.13	.43	5.70	.08	7.0	93.0	....
.51	19.76	.01	79.72	.03	19.76	74.62	.48	5.10	.01	8.6	91.4	....
.35	20.14	.00	79.51	.03	20.14	76.05	.32	3.45	.01	8.5	91.5	....
.35	20.24	.02	79.39	.03	20.24	76.43	.32	2.96	.02	9.8	90.2	....
.40	20.13	.00	79.47	.03	20.13	76.01	.37	3.46	.00	9.7	90.3	....
.20	20.43	.03	79.34	.03	20.43	77.15	.17	2.19	.03	7.2	92.8	....

MINE IN EASTERN PENNSYLVANIA.

0.09	20.80	0.05	79.06	0.03	20.80	78.55	0.06	0.51	0.05	10.5	89.5	....
.11	20.74	.18	78.97	.03	20.74	78.32	.08	.65	.18	11.0	89.0	....
.15	20.62	.36	78.87	.03	20.62	77.86	.12	1.01	.36	10.6	89.4	....
.12	20.68	.19	79.01	.03	20.68	78.09	.09	.92	.19	8.9	91.1	....
.13	20.69	.24	78.94	.03	20.69	78.13	.10	.81	.24	11.0	89.0	....
.13	20.65	.34	78.88	.03	20.65	77.98	.10	.90	.34	10.0	90.0	....
.16	20.48	.65	78.71	.03	20.48	77.34	.13	.65	.65	8.7	91.3	....
.09	20.52	.80	78.59	.03	20.52	77.49	.06	.80	.80	5.0	95.0	....
.16	20.49	.94	78.41	.03	20.49	77.37	.13	1.04	.94	11.0	89.0	....
.19	20.37	1.04	78.40	.03	20.37	76.92	.16	1.48	1.04	9.8	90.2	....
.20	20.29	1.17	78.34	.03	20.29	76.62	.17	1.72	.24	9.0	91.0	....
.22	20.39	1.01	78.38	.03	20.39	77.00	.19	1.38	1.01	12.3	87.7	....

TABLE 5.—*Results of analyses of*

SAMPLES IN SERIES 5,

Bottle No.	Sample No.	Distance of point of sampling from intake shaft.	Point of sampling.	Quantity of air per minute.	Temperature at point of sampling. Wet bulb.	Dry bulb.	Relative humidity.
		Feet.		*Cubic feet.*	°*F.*	°*F.*	*Per ct.*
1	6328	50	Main east entry at air course	10,100	66	67	95
2	6322	562	Main east entry	15,000	66	67	95
3	6321	1,300	.....do	13,680	66	67	95
4	6331	2,060	.....do	7,980	66	67	95
5	6329	2,600	.....do	7,790	65	66	95
6	6330	3,560	No. 1 room, off second west entry	Air still.	66.5	67	97
7	6327	4,025	Room 2, off second west entry, off main east entry	Air still.	65	66	95
8	6326	4,175	Room 7, second west entry, off main east entry	Air still.	66	67	95
9	6323	4,400	Room 9, second west entry, off main east entry, at face	Air still.	67	68	98
10	6324	5,100	Second west air course at overcast on first east entry	1,470	66	67	95
11	6325	6,500	Return at junction of main east entry and main north entry	16,000	67	68	95

SAMPLES IN SERIES 6,

Bottle No.	Sample No.	Distance of point of sampling from intake shaft.	Point of sampling.	Quantity of air per minute.	Wet bulb.	Dry bulb.	Relative humidity.
1	6303	(a)	Foot of air shaft, intake air	35,700	65	66	95
2	6305	2,000	Fourth west air course	9,150	60	61	94
3	6306	3,000	.....do	13,500	61	62	94
4	6307	4,000	.....do	11,425	65	66	95
5	6308	5,300	Face of room in eighth north entry, off fourth west air course	Air still.	63	63	100
6	6309	6,300	Face of eighth north entry, off fourth west air course	7,200	64	65	95
7	6310	6,900	Room 9, first south entry, off third west air course	Air still.	66	67	95
8	6311	7,000	Room 1, at face, off first south entry, off third west air course	Air still.	65	66	95
9	6312	8,100	Room 2, near face at breakthrough, off eighth north entry, off fourth west air course	8,710	65	66	95
10	6313	9,800	Fourth west air course, return side	8,710	65	66	95
11	6314	10,700	.....do	13,000	66	67	95
12	6315	11,600	.....do	15,570	65	66	95
13	6316	13,000	.....do	16,575	65	66	95

a Sample taken at foot of shaft.

mine-air samples, series 2 to 6—Continued.

FROM A MINE IN INDIANA.

Composition of sample as received.				Recalculated analysis to show air, black damp, and methane.						Composition of black damp.		Specific gravity (air=1).
				Air.			Black damp.		Methane.			
CO_2.	O_2.	CH_4.	N_2.	CO_2.	O_2.	N_2.	CO_2.	N_2	CH_4.	CO_2.	N_2.	
P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
0.05	20.87	0.00	79.08	0.03	20.87	78.81	0.02	0.27	0.00	6.9	93.1	....
.08	20.81	.00	79.11	.03	20.81	78.59	.05	.52	.00	8.8	91.2	....
.06	20.82	.00	79.12	.03	20.82	78.62	.03	.50	.00	5.7	94.3	....
.09	20.80	.01	79.10	.03	20.80	78.54	.06	.56	.01	9.7	90.3	....
.19	20.55	.00	79.26	.03	20.55	77.60	.16	1.66	.00	8.8	91.2	....
.56	19.55	.01	79.88	.03	19.55	73.82	.53	6.06	.00	8.0	92.0	....
.76	19.04	.07	80.13	.03	19.04	71.90	.73	8.23	.07	8.2	91.8	....
.74	19.00	.07	80.19	.03	19.00	71.74	.71	8.45	.07	7.8	92.2	....
.85	18.67	.13	80.35	.03	18.67	70.50	.82	9.85	.13	7.7	92.3	....
.54	19.53	.04	79.89	.03	19.53	73.75	.51	6.14	.04	7.7	92.3	....
.55	19.59	.02	79.84	.03	19.59	73.97	.52	5.87	.02	8.1	91.2	....

FROM A MINE IN INDIANA.

0.04	20.90	0.00	79.06	0.04	20.90	79.06	0.00	0.00	0.00	0.0	0.0	....
07	20.78	.10	79.05	.03	20.78	78.47	.04	.58	.10	6.5	93.5	....
.11	20.70	.10	79.09	.03	20.70	78.17	.08	.92	.10	8.0	92.0	....
.14	20.67	.13	79.06	.03	20.67	78.05	.11	1.01	.13	9.8	90.2	....
.25	20.43	.41	78.91	.03	20.43	77.15	.22	1.76	.41	11.1	88.9	....
.21	20.43	.22	79.14	.03	20.43	77.15	.18	1.99	.22	8.3	91.7	....
.25	20.31	.26	79.18	.03	20.31	76.69	.22	2.49	.26	8.1	91.9	....
.49	19.89	.52	79.10	.03	19.89	75.11	.46	3.99	.52	13.9	86.1	....
.24	20.57	.14	79.05	.03	20.57	77.68	.21	1.37	.14	13.3	86.7	....
.37	20.36	.29	78.98	.03	20.36	76.88	.34	2.10	.29	13.9	86.1	....
.76	19.58	.18	79.48	.03	19.58	73.94	.73	5.54	.18	11.6	88.4	....
.55	19.67	.16	79.62	.03	19.67	74.28	.52	5.34	.16	8.9	91.1	....
1.27	18.08	.12	80.53	.03	18.08	68.27	1.24	12.26	.12	9.2	90.8	....

COMMENT ON RESULTS OF SAMPLING AND ANALYSIS.

SAMPLES IN SERIES 2.

It will be noted that the first sample of series 2 was collected 2,475 feet from the intake, and the succeeding samples at greater distances, until the last one was taken 6,575 feet from the intake. In general, the first samples were lower in carbon dioxide content and higher in oxygen content, and consequently contained less black damp, but the change was not so marked as in the samples of other series. The composition of the black damp was fairly constant, ranging between 4.4 per cent and 12.5 per cent carbon dioxide and between 87.5 and 95.6 per cent nitrogen. The average composition of the black damp was carbon dioxide, 8.4 per cent; nitrogen, 91.6 per cent.

SAMPLES IN SERIES 3.

The first eight of the samples in series 3 were collected in still air at working faces. The proportion of carbon dioxide ranged from 0.20 to 0.67 per cent, the oxygen content ranged from 19.54 to 20.43 per cent, and the percentage of black damp ranged from 5.75 to 6.13 per cent, a small but still noticeable increase. The last five samples were collected in the air current and the greater purity of the air is noticeable. The composition of the black damp was remarkably constant, ranging from 6.7 to 9.8 per cent carbon dioxide and from 90.2 to 93 per cent nitrogen. The average composition of the black damp was carbon dioxide, 8.4 per cent; nitrogen, 91.6 per cent.

SAMPLES IN SERIES 4.

The change in the composition of the air as it traversed the mine is much better indicated by the data regarding the samples in series 4 than by those for the samples in the two preceding series, because the volume of air traversing the place where each sample was collected did not differ enough to influence markedly the percentage change in the carbon dioxide and oxygen contents. Some irregularities due to this cause were, however, noticeable. At the place where sample 6294 was collected, 15,300 cubic feet of air was passing, whereas at the place where sample 6299 was collected, 7,170 cubic feet of air was passing; this difference in the quantity of air passing made the percentage of black damp in sample 6294 (1.57) appear less, whereas in reality the cubic feet of black damp was greater, being 240 cubic feet (15,300 by 1.57) as against 136 cubic feet (7,170 by 1.89) for sample 6299.

SAMPLES IN SERIES 5.

A noticeable increase in the percentage of black damp is seen in the samples in series 5. Samples 6330, 6327, 6326, and 6323 were collected at working faces in still air, and consequently were much higher in black

damp than the other samples. These samples also showed a progressive increase in black damp content as the distance from the intake was increased. The greatest vitiation in the moving air was noticeable in samples 6324 and 6325.

The composition of the black damp was nearly constant. The average proportion of carbon dioxide was 9.8 per cent, and of nitrogen 90.2 per cent.

SAMPLES IN SERIES 6.

Progressive vitiation of the air is also noticeable in the samples of series 6. A noticeable feature is the large increase of black damp in sample 6316, as compared with the proportion in sample 6315. The average percentage of carbon dioxide in the black damp is 10.2 per cent and of nitrogen 89.8 per cent.

EFFECTS OF TEMPERATURE AND HUMIDITY AND OTHER FACTORS.

The temperatures in the mine where the samples of series 2 were collected were all moderate, in fact, highly desirable working temperatures. The carbon dioxide content was not high, but the oxygen content was satisfactory and practically no methane was present. No fault could be found with the condition of the air in this mine.

In the mine where the samples of series 3 were collected the carbon dioxide content was higher than in the foregoing mine, the oxygen content lower, and the temperatures a little higher. However, according to modern ideas of the purity of air and the effects of mine air on individuals, one would say the conditions of the air in this mine were all right.

The same is true of the mine where the samples of series 4 were collected, except that the ventilation needed closer watching because of the presence of methane in the air current. A proportion of 1.01 per cent of methane in an airway (sample 6294, where 15,300 cubic feet of air was passing) is rather high. However, the sample represented an anthracite mine in which dust does not constitute the menace that it does in bituminous mines. Many of these mines are extremely gaseous; consequently more difficulty is experienced in keeping the methane down to small percentages than in most bituminous mines.

In the mine where the samples of series 5 were collected conditions were fairly good. At one working face (sample 6323) the oxygen content was 18.67 per cent and the carbon dioxide content 0.85 per cent. The mine was not gaseous. The temperatures were satisfactory.

In the mine where the samples in series 6 were collected, one sample (No. 6316) was high in carbon dioxide (1.27 per cent) and

low in oxygen (18.08 per cent). This sample was collected in the return air where 16,575 cubic feet of air was passing. The temperatures were satisfactory.

EFFECTS OF LEAKAGE OF AIR THROUGH DOORS, BRATTICES, ETC.

Necessarily the composition of the air in an entry is influenced greatly by the leakage of air through stoppings, doors, brattices, and overcasts. For instance, the air may be practically normal at a particular place in an entry, not having traveled very far in the mine, and hence not having reacted with much coal. A short distance farther, however, a decided change in its carbon dioxide and oxygen content might be caused by badly vitiated air leaking through a door. That mine will have the tightest doors, overcasts, etc., in which the air shows a gradual and uniform change in composition as it travels through the mine.

INFLAMMABLE GAS IN MINES.

Although this report deals primarily with the vitiation of coal-mine air by changes in the content of carbon dioxide and of oxygen, some reference to the presence of inflammable gas (methane) is required, because coal-mine ventilation is largely governed by the quantity of inflammable gas entering the workings.

For instance, the bituminous-mine law of Pennsylvania states that in a nongaseous mine the minimum quantity of air shall be not less than 150 cubic feet per minute for each person employed, and in a mine wherein explosive gas is being generated in such quantities that it can be detected by an approved safety lamp, the minimum quantity of air shall be not less than 200 cubic feet per minute for each person employed therein, and as much more in either case as one or more of the inspectors may deem requisite. Further, under the law mentioned, the use of open lights is prohibited in any entry, airway, traveling way, room, or any other working place where explosive gas is likely to be encountered, and in all such places locked safety lamps must be used exclusively.

In other words, the fact that coal mines generate inflammable gas in greater or less quantities has necessitated the introduction of fresh air to dilute the gas, and this natural gaseous condition of many coal mines has had a great deal to do with keeping the air of coal mines in good condition. The presence of inflammable gas introduces the danger of disastrous explosions, especially in mines where the air is not controlled properly, but this danger, by leading to the continuous introduction of immense volumes of fresh air, year after year, has undoubtedly been the chief element in assuring good ventilation.

In general the air in mines that are termed "gaseous," and consequently are required to have large volumes of air coursing through them and must be inspected more thoroughly to keep all parts of the workings free of inflammable gas, is higher in oxygen, lower in carbon dioxide, lower in temperature, and moves faster, and in all ways has a higher standard of purity than the air in those mines that are termed nongaseous and hence are not required by law to have as much fresh air passing through them, and, in which, because they are not menaced by inflammable gas, the ventilation is not regulated as carefully.

OCCURRENCE AND COMPOSITION OF GAS IN ROCK STRATA OF THE CRIPPLE CREEK GOLD-MINING DISTRICT, COLORADO.

The senior authors of this paper, while in Colorado on other official business, made a trip to the Cripple Creek gold-mining district to get more data than is at present available regarding the composition of the gas that issues from the rock strata into the gold mines of the district. This gas was said to cause suffocation, and hence to be a menace to life. The data presented herein relate principally to the composition of this gas and its effects on men and on lights.

It is estimated that 25 to 100 miners have been killed by the rock gas in the 25 years that mining has been vigorously carried on at Cripple Creek. Many men have had narrow escapes from death, some of them having been incapacitated for days. In addition, on certain days men can not enter some of the workings. At a few mines fans are used to force outside air into the mines, thus improving conditions in large measure, but even this plan, as used at Cripple Creek, is not entirely adequate at all times.

ORIGIN OF THE STRATA GAS.

Lingren and Ransome [a] believed that the gas found in the rocks of the Cripple Creek mining district represents the last exhalations of the extinct Cripple Creek volcano. In support of this they stated that little timbering is used in the mines, hence one can not account for the decrease of oxygen or increase of carbon dioxide by oxidation of timber. Further, there is only a small proportion of pyrite and carbonates present in the rocks and ores as compared to that in many other mines entirely free from gas. Moreover, no gas occurs in the oxidized zone, showing that oxidation can not have anything to do with it, and as the gas increases with depth it must be mainly accumulated in strata at depths below those penetrated by the deepest mines of this district.

[a] Lingren, W., and Ransome, F. L., Geology and gold deposits of the Cripple Creek mining district: U. S. Geol. Survey, Prof. Paper 54, 1906, p. 257.

EFFECTS OF BAROMETRIC PRESSURE ON OUTFLOW OF STRATA GAS.

The gas is confined in the rock under such low pressure that variations in the outside atmospheric pressure may materially affect the outflow of gas into the mines. Also, at these mines where fans force air in, thereby putting the workings under about 6 or 7 inches of water pressure (about 0.5 inch of mercury), the workings are more or less free from the rock gas which is held back in the pores and crevices of the rocks. The variation in outside atmospheric pressure is easily more than 0.5 inch of mercury, and the effect of the barometric pressure on the outflow of gas is appreciated by mining men.

COLLECTION OF SAMPLES OF GAS IN FOUR METAL MINES.

One of the authors collected samples of gas in the Mary McKinney, Anaconda, Midget, and Cresson mines. Data regarding the samples are contained in Table 8. General remarks on some of the mines represented in the table are presented as bearing on the data given in the table.

OBSERVATIONS ON GAS IN MIDGET MINE.

The Midget mine is situated on the west slope of Gold Hill. The workings are partly in breccia and partly in the fine-grained gray gneiss common to the district. The mine has 10 levels. From the shaft to the tenth level is about 900 feet. About 20 men work underground on a leasing system. As regards the men in the levels where samples of gas were collected, four men were in the seventh level, four men on the eighth, and nobody, on account of bad gas conditions, on the ninth or tenth. In the afternoon of the day of sampling the men had to leave the eighth level on account of the gas. Gas conditions were said to be worse than usual at the mine on that day.

The Midget mine uses a pressure system of ventilation. On the surface is a 5-foot Sturtevant fan driven by a 20-horsepower motor which forces air through an air compartment in the shaft to a point below the second level, whence the air spreads into the various levels. In the drifts at various distances from the shaft are air doors, built of 1-inch boards and tightened with canvas. Thus, a small pressure, 3 or 4 inches water gage, is placed on the mine workings, sufficient to check in part the outflow of gas from the rocks and enable the men to do more work than they could without the fan. But when the outflow of gas is stronger than usual, workmen are sometimes driven from the lower levels, and occasionally can not reenter for days.

The Conundrum mine was the first in the Cripple Creek district to install this system of ventilation. This mine adjoins the Midget.

Of the series of samples collected at the Midget mine, only one, No. 664, was collected outside the pressure area or zone and beyond the door. Candles burned freely inside the door, but 20 feet beyond they went out, indicating at this point less than 17 per cent of oxygen. When the door was opened the air rushed through with sufficient strength to blow out the lights.

GENERAL OBSERVATIONS REGARDING GASEOUS CONDITIONS IN ANACONDA MINE.

The workings of the Anaconda mine in the town of Anaconda extend through Gold Hill. The mine is worked through an adit having a portal at Anaconda. The prevailing country rock is breccia, with some irregular bodies of latite-phonolite and a few dikes of phonolite and basalt.

The Anaconda shaft is reached by means of an adit 1,135 feet long. The fourth and fifth and lower levels are at times much troubled by rock gas. The progressive vitiation of the air in the fourth drift of the fourth level is shown by the data presented in Table 6 regarding the first six samples from the Anaconda mine. In taking sample 691, 300 feet north of the shaft, which contained 9.02 per cent of oxygen and 5.51 per cent of carbon dioxide, the sampler advanced with an acetylene lamp held near his mouth, until the lamp went out (about 12 or 13 per cent oxygen), then reached down and collected the sample near the floor. The dangerous stratification of the gas is well shown by these analyses. At the height of a man's mouth the air was breathable, but at the floor a man would collapse quickly because of the small amount of oxygen (5.51 per cent) there. Similar stratification is shown by samples taken at other points.

It was noticed on another occasion that a lighted candle would burn fairly well at one point in a drift, but become extinguished further along, although great care was taken that it should not be extinguished by a sudden movement or jerk. At a point still further in the drift it would stay lighted. Usually, though, the nearer one approached the breast of a drift containing much rock gas the worse the air became.

OBSERVATIONS ON GAS IN CRESSON MINE.

The Cresson mine is on Raven Hill about two and one-half miles from the Midget mine and two miles from the Anaconda mine. It has 13 levels. The prevailing country rock is andesite breccia.

A wooden door made as tight as possible is placed in each drift beyond the workings and a compressed-air pipe is run through a

hole in the door and is sealed around as tightly as possible. A valve placed inside the door provides for turning on the compressed air. When the air is turned on the rock gas is kept back. The effectiveness of this plan is shown by the analyses of samples collected inside and beyond one of these doors.

In collecting sample 760 the collector advanced 10 feet beyond where the air was comparatively good, holding his breath, then quickly snapped the sealed and evacuated glass sample bottle, whereupon the mine gas immediately filled the bottle, after which he quickly came out. Even so he had a narrow escape, for his knees became weak and his mind slightly hazy, owing no doubt to inadvertently breathing a little of the atmosphere. As soon as he came to better air 10 feet away he felt all right.

Sample 760 showed the effect of the compressed air in holding back the rock gas. At the height of a man's head just at the door the oxygen content of the atmosphere was 16.23 per cent, whereas 10 feet beyond the door the oxygen content was only 2.69 per cent. This sample more closely approximated the pure rock gas than any other collected.

So-called blowers of gas are difficult to find in the Cripple Creek mines. The escape of gas from the rocks usually is uniform from thousands of small pores rather than from large outbursts at one particular place. If one calculates the air-free composition of this sample (No. 760), by eliminating the nitrogen and oxygen according to the proportions found in atmospheric air, the composition becomes 12.69 per cent carbon dioxide and 87.31 per cent nitrogen. These figures show the composition of the pure rock gas, if one assumes that the air in the sample as collected was due to dilution of the rock gas by the mine air.

Apropos of the occasional small outbursts of gas from the rocks, it is interesting to mention at this point an incident related by the superintendent of the Midget mine. In the course of his duties in the mine he was leaning against the wall of a drift with his face close to the rock, talking to another man. Suddenly he felt dizzy and breathless and his knees became weak. Although not knowing the cause of his distress he changed his position and soon felt better. Then it occurred to him that possibly a small feeder of gas was finding its exit close to where he had been standing; so he put his carbide lamp up against the rock at that place. It was immediately extinguished, showing that his assumption had been correct.

TABULATED DATA.

Table 6, presenting detailed data regarding the samples taken in the four mines under consideration, follows.

TABLE 6.—*Data on gas samples collected in four mines in the Cripple Creek district.*

MIDGET MINE.

Sample No.	Place of taking sample.	Bed.	Temperature. Wet bulb.	Temperature. Dry bulb.	Relative humidity.	Barometer.	Candle burned.	Acetylene lamp burned.	CO_2.	O_2.	Combustible gas.	N_2.	Total.	Remarks.
			°F.	°F.	P. ct.	Inches.			P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	
781	Eighth level, north drift	Midget Murray vein.	73	76	88	21.6	No	Yes	3.66	16.57	0.00	79.77	100	--
776	Eighth level, in Olson-Hollengrain stope.						Yes	Yes	1.00	18.92	.00	80.08	100	
663	Eighth level, at breast	Midget Murray vein.	60	65	75		Yes	Yes	.98	20.08	.00	79.04	100	
770	Raise, 40 feet from breast	Intermediate vein.	63.5	69	76		No	No [a]	8.84	10.06	Trace.[b]	80.30	100	
286	Ninth level, north drift, 250 feet from shaft.	Conundrum vein	63	69	76	21.66	Yes [c]		2.08	18.53	.00	79.39	100	No men working on ninth level.
769	In shaft at ninth level, 830 feet from surface.		61	66	74				.11	20.79	.00	79.10	100	Air in shaft would not move vanes of anemometer.
400	Seventh level, 375 feet northwest of shaft.	Intermediate vein.	63.5	74	59			No [d]	8.68	10.86	.00	80.46	100	No men working here.
784	Seventh level, 250 feet north of shaft, 75 feet from breast.	Midget Murray vein.						No	7.35	11.63	.00	81.02	100	Do.
684	Sixth level, outside of pressure zone, about 200 feet beyond air line.		50.5	54	61				5.09	15.06	.00	79.85	100	

[a] Would burn 1 foot above where sample was collected.
[b] Less than 0.02 per cent.
[c] Feebly.
[d] Burned 2 feet above where sample was taken.

TABLE 6.—*Data on gas samples collected in four mines in the Cripple Creek district*—Continued.

ANACONDA MINE.

Sample No.	Place of taking sample.	Bed.	Temperature. Wet bulb.	Temperature. Dry bulb.	Relative humidity.	Barometer.	Candle burned.	Acetylene lamp burned.	CO_2.	O_2.	Combustible gas.	N_2.	Total.	Remarks.
			°F.	°F.	P. ct.	Inches.			P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	
691	Fourth level, fourth drift, 300 feet north of shaft.	....	56	59	85	....	....	(a)	9.02	5.51	Trace.[b]	85.47	100	
986	Fourth level, fourth drift, 275 feet from shaft.	....	56	59	85	21.6	....	....	7.96	7.50	.02	84.52	100	Nobody working on this level; influx of gas in the afternoons and about one-third of the time in the morning; generally prevents work in this level.
937	Fourth l drift, 250 feet from s	....	....	....	....	....	....	No[c]..	5.43	11.78	.00	82.79	100	
750	Fourth l drift, 150 feet from a oor.	....	....	....	....	....	....	No....	8.09	7.19	Trace.[b]	84.72	100	
104	Fourth l drift, at roof, 7 feet f	....	....	....	....	....	....	....	1.64	18.30	.00	80.06	100	
948	Fourth drift, 5½ feet from floor.	....	....	....	....	....	Yes[d]..	....	1.51	18.44	.00	80.05	100	
690	Fifth level, at breast of north drift, 200 feet below fourth level.	....	64	68	82	....	No....	Yes...	1.69	17.70	.00	80.61	100	
747	Fifth level, at foot of shaft........	....	57.5	60.5	85	....	Yes...	....	.32	20.46	.00	79.22	100	

MARY McKINNEY MINE.

Sample No.	Place of taking sample.	Bed.	Temperature. Wet bulb.	Temperature. Dry bulb.	Relative humidity.	Barometer.	Candle burned.	Acetylene lamp burned.	CO_2.	O_2.	Combustible gas.	N_2.	Total.	Remarks.
761	feet from up about drift and shaft.	No. 2 vein.......	56	60	80	21.87	....	....	1.82	10.70	.00	87.48	100	Nobody was working in this raise.
762	north of orth drift,	....	58.5	61	86	....	....	No[c]..	7.54	8.01	.00	84.45	100	Nobody was working at this place; the gas sometimes fills entire drift; conditions were worse than usual at time of sampling.

780	Eighth level, at roof, No. 12 north drift, at junction of Black vein drift.	....	....	....	....	....	....	....	Yes...	4.17	13.87	Trace.[b]	81.96	100	
808	Junction of No. 2 vein and main crosscut from shaft, 1,250 feet from shaft.[f]	No vein.	....	54.5	58	82	....	....	....	.25	20.50	.02	79.23	.100	
	CRESSON MINE.														
797	...oor, ...r.	....	....	....	60	....	....	....	No....	5.90	7.03	.00	84.07	100	Door was open slightly when sample was collected.
796	...oor,	....	....	....	....	....	....	....	Yes...	2.85	16.23	.00	80.92	100	
760	...t be-	....	....	....	....	....	....	....	....	11.08	2.69	.03	86.20	100	See pp. 60, 62 for extended comments.
759	...ope,	....	....	60	64	....	....	Yes...	....	2.35	17.19	.00	80.46	100	Sample collected 100 feet inside door where samples 797, 796, and 760 were collected.
771	Eleventh level, at the shaft......	....	....	64.5	65.3	81	....	....	....	.29	20.56	.00	79.15	100	
754	Seventh level, 700 feet from shaft, near bottom of drift.	....	....	61	64	86	....	No....	Yes...	1.01	14.05	Trace.[b]	84.94	100	
753	Seventh level, 700 feet from shaft,	....	....	61	64	86	....	No....	Yes...	.60	16.74	.00	82.66	100	
772	..., at center of drift.	....	....	61	64	86	....	No....	Yes...	.62	16.65	.00	82.73	100	
943	Seventh level, 600 feet from shaft..	....	....	....	....	....	....	....	....	.39	20.03	.00	79.58	100	
944	Seventh level, 650 feet from shaft..	....	....	....	....	....	....	....	....	.51	17.78	.00	81.71	100	

[a] Carbide lamp went out ... it 5½ feet above floor.
[b] Result corresponds with ..., which showed that acetylene lamp is extinguished when oxygen content of atmosphere drops to about 12 or 13 per cent
[c] Less than 0.02 per cent.
[d] Faulty.
[e] Would burn at roof.
[f] Shaft was acting as a downcast, and sample was collected 250 feet from upcast; air seemed good.

SPECIAL AIR SAMPLES FROM MARY M'KINNEY MINE.

The Mary McKinney mine is situated on the south side of Squaw Gulch, opposite the town of Anaconda. The first ore was shipped in 1893. The workings are mainly in breccia and phonolite. Data on the air samples collected in this mine by one of the authors is shown in Table 8.

At the instance of the authors mine-air samples were collected by A. G. Suydam, a mining engineer of Cripple Creek, for 22 days, almost daily, in the Mary McKinney mine, in the No. 12 north drift, on the 800-foot level, about 1,800 feet from the shaft. The point of sampling was close to a fissure in the rock from which a "feeder" of rock gas intermittently issued.

Table 7 following shows the number of samples collected at this point by Mr. Suydam, date of sampling, results of analyses, and the direction of the wind and the barometric pressure on the date the samples were collected:

TABLE 7.—*Data regarding gas samples collected near feeder in Mary McKinney mine.*

Sample No.	Date of collection of sample.	Analysis.			Direction of wind.	Barometric pressure.	Condition of weather.	Strength of wind.
		CO_2	O_2	N_2				
	1915.					In. Hg.		
6987	Nov. 1	0.24	20.58	79.18	N. to NW	22.25	Clear	Mild.
6984	Nov. 3	12.06	2.97	84.97	SW	22.26	do	Do.
6986	Nov. 5	13.27	.94	85.79	S. to SW	22.16	Partly cloudy	Do.
6981	Nov. 9	8.84	7.83	83.33	S. to SW	22.06	do	Do.
6983	Nov. 11	.17	20.75	79.08	NW. to N	21.96	Clear and cold	Strong.
6980	Nov. 13	11.05	3.39	85.56	S. to SW	21.93	Partly cloudy	Mild.
6988	Nov. 15	7.86	8.89	83.25	N. to NW	22.00	Clear and cold	Brisk.
6989	Nov. 22	.41	20.64	78.95	N. to NW	22.16	Clear	Mild.

In addition to the results given above some interesting observations were made by Mr. Suydam as to the effect of different wind and barometer conditions on the air in the drift of the Mary McKinney mine as indicated by candles and acetylene lights. These data are shown in Table 8, following.

TABLE 8.—*Data regarding effect of wind and barometer conditions on air in drift of Mary McKinney mine.*

Date.	Barometer.	Wind.		Sky.	Remarks.
		Direction.	Strength.		
1915.					
Oct. 11	21.875	S. to S. W	Mild	Cloudy	Mine closed because of strata gas.
12	21.955	N. to N. E	Calm	Clear	Air good in drift.
14	21.985	N. E. to N	do	Cloudy	Some strata gas in drift.
15	22.030	N. E. to N	Strong	do	Air good.
16	22.130	S. W. to S	Calm	do	Do.
17	22.185	N. to N. W	Mild	Clear	Do.
19	22.175	do	do	do	Do.
21	22.240	S. to S. E	do	do	Do.
22	22.350	do	Calm	do	Some strata gas in drift.
23	22.295	do	do	do	Do.
25	22.200	N. to N. W	do	do	Air good.
26	22.290	N. to N. E	Light	do	Do.
27	22.380	N. to N. W	Calm	do	Do.
28	22.385	do	do	do	Do.
30	22.235	S. W. to S. to W	Brisk	Partly cloudy.	Strata gas in drift.
Nov. 2	22.260	W. to S. W. to N. W.	Mild	do	Air good.
10	21.750	S. to S. W	do	Fog	Do.
16	21.800	N. to N. W	do	Clear (cold)	Strata gas in drift.
18		do	do	do	No gas; air good.
19	22.185	N. W. to N	do	Clear	Air good.

Regarding those samples that were collected and analyzed, the total number of samples collected was too small to permit the drawing of rigid conclusions. With one exception (sample No. 6988) the largest amounts of carbon dioxide and the smallest amounts of oxygen were found in samples collected when the wind was from the south or southwest. This relation agrees with statements of some mining men that when the wind came from the south or southwest the most rock gas entered the mines.

Table 7 shows that rock gas was present in the north drift of the Mary McKinney mine when the wind was south or southwest in 4 cases and when it was north in 2 cases. Also in 10 cases the air was good in the drift when the wind was from the north, and in 3 cases when it was from the south or southwest. Also a consistent relation could not be traced between the barometric pressure and the presence of rock gas.

Some of the mining men of the Cripple Creek district have a theory that when the wind comes from the south or southwest it sweeps up canyons where the rock outcrops. These, being somewhat porous, permit the entrance of atmospheric air, which forces the rock gas into the mine workings.

COMPOSITION OF STRATA GAS SAMPLES CALCULATED ON AIR-FREE BASIS.

It was impossible to procure samples of pure strata gas as it issued from the rocks in the Cripple Creek mines during the visit of one of the authors. Entrance was made as far as it was possible to penetrate into some of the drifts that were most affected. A sample (No. 760) containing 2.69 per cent oxygen was obtained from the Cresson mine and was the sample containing the largest percentage of strata gas. Undoubtedly if one had been able to penetrate 15 or 20 feet farther into the drift a sample practically devoid of oxygen could have been procured. One can determine rather closely the composition of the strata gas, however, by selecting those samples that contained the smallest percentages of oxygen and calculating them air-free, as has been done in the following table:

Air-free composition of samples containing small percentages of oxygen.

MIDGET MINE.

Sample No.	CO_2.	O_2.	Combustible gas.	N_2.	Total.	Calculated air-free.	
						CO_2.	N_2.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
770	8.84	10.86	0.00	80.30	100.00	18.37	81.63
400	8.68	10.86	.00	80.46	100.00	18.03	81.97
794	7.35	11.63	.00	81.02	100.00	16.53	83.47

ANACONDA MINE.

966	7.96	7.50	0.00	84.54	100.00	12.40	87.60
937	5.43	11.78	.00	82.79	100.00	12.41	87.59
691	9.02	5.51	.00	85.47	100.00	12.23	87.77
750	8.09	7.19	.00	84.72	100.00	12.32	87.68

MARY McKINNEY MINE.

762	7.54	8.01	0.00	84.45	100.00	12.20	87.8
976	5.88	10.71	.00	83.41	100.00	12.03	87.97

CRESSON MINE.

797	8.90	7.03	0.00	84.07	100.00	13.4	86.6
760	11.08	2.69	.00	86.23	100.00	12.71	87.29

These calculations show that the composition of the strata gas varied between 12.03 and 18.37 per cent of carbon dioxide and 81.63 and 87.97 per cent of nitrogen. The average of all of the results is 13.87 per cent of carbon dioxide and 86.13 per cent of nitrogen. Thus, the rock gas is a mixture of carbon dioxide and nitrogen. In making the calculations it was assumed that no oxygen is present

in the gas as it issues from the rock, and that any air in the samples was due to dilution of the rock-strata gas by the air of the mine. This assumption appears to be justified because in a drift more or less filled with rock-strata gas the gas becomes less and less diluted with oxygen as one travels from the good air of the mine farther and farther into the bad air of the drift.

A tendency was noticed among the mining men of Cripple Creek to speak of the strata gas as carbon dioxide or at least as if carbon dioxide was the predominating constituent. This is not the case. Nitrogen is much in excess. The bad effects produced are principally due to the fact that the rock gas so dilutes the air of the mines that the oxygen falls to a point where lights will not burn or so low that life is endangered.

In the authors' opinion the acetylene light should not be used as the sole warning against the presence of gas in these mines. It is true that where the acetylene lamp burns there is enough oxygen (12 to 13 per cent) in the air to support life, but under such conditions the air only a short distance beyond in a drift or at the floor may be fatal to life. The warning of a candle flame affords a much wider margin of safety. At some mines the operators allow no work to be performed where a candle will not burn.

EFFECT ON MEN OF PARTIAL PRESSURES OF OXYGEN.

The effects of carbon dioxide and oxygen on men and lights have been discussed in a previous part of this report. A point that can be profitably emphasized here is the effect of the partial pressure of the oxygen on men.

The effects of working in bad air of the Cripple Creek mines are typical of the effects produced by any low oxygen and high carbon dioxide atmospheres. After a day's work the men suffer a feeling of oppression, heaviness, and lassitude, or sleepiness, and a loss of appetite, the degree of distress depending, of course, upon the extent of vitiation of the air. When the air gets very bad, say when a candle will not burn, slight exertion causes breathlessness. Much exposure in bad air brings on headaches and nausea and complete exhaustion. That more fatalities do not occur is due to the fact that the men fairly well appreciate the warning of their lamps and are careful about venturing where acetylene lamps will not burn. Collapse may be very sudden in atmospheres low in oxygen. In fact, it is typical of such atmospheres that little warning is given of their great danger. In some cases men who have collapsed and been rescued have been days recovering. The after effects are very similar to those produced by poisoning by carbon monoxide.

COMPARISON BETWEEN THE INDICATIONS AFFORDED BY CANDLE AND BY ACETYLENE FLAME AND ANALYSES OF THE SAMPLES.

At each place where samples were collected by one of the authors he made a note of the condition of his lamp flames, both candle and acetylene. It is interesting to compare these observations with the oxygen percentages of the various samples, as in the following table:

Data showing relation of oxygen contents of samples to burning of candle and acetylene lamp flames.

Sample No.	Oxygen in sample.	Candle burned.	Acetylene lamp burned.	Sample No.	Oxygen in sample.	Candle burned.	Acetylene lamp burned.
MIDGET MINE.							
	Per cent.				*Per cent.*		
781	16.57	No	No.	400	10.86	No	No.
776	18.92	Yes	Yes.	794	11.63	No	No.
770	10.86	No	No.	664	15.06	No	Yes.
296	18.53	Yes [a]	Yes.				
ANACONDA MINE.							
966	7.50	No	No.	694	18.30	Yes [a]	Yes.
937	11.78	No	No.	948	18.44	Yes [a]	Yes.
691	5.51	No	No.	690	17.70	Yes [a]	Yes.
750	7.19	No	No.				
MARY McKINNEY MINE.							
761	10.70	No	No.	780	13.87	No	Yes.
762	8.01	No	No.	976	10.71	No	No.
CRESSON MINE.							
797	7.03	No	No.	754	14.05	No	Yes.
795	16.23	No	Yes.	753	16.74	No	Yes.
760	21.69	No	No.	772	16.65	No	Yes.
759	17.19	Yes [a]	Yes.	944	17.78	Yes [a]	Yes.

[a] Feebly.

In general these results show that the candle flame became extinguished when the oxygen in the atmosphere fell to 17 to 18 per cent, and that the carbide flame was extinguished when the oxygen content fell to 12 to 14 per cent.

COMBUSTIBLE GAS IN THE ROCK-STRATA GAS.

Eight of the samples examined contained traces of combustible gas. The largest proportion was 0.03 per cent. Presumably all of the samples contained small proportions that could not be detected by analysis. The relation between the contraction and the carbon dioxide content indicated that this combustible gas was methane.

Rarely a small outburst of gas is encountered that burns when a torch is applied to it, but quickly burns itself out.

VENTILATION CONDITIONS OF 15 METAL MINES AS INDICATED BY ANALYSES OF AIR SAMPLES.

In Table 9 following are shown data regarding samples of air collected in 15 metal mines. Most of the samples were collected by H. M. Wolflin, mining engineer of the Bureau of Mines. Some were collected by Edwin Higgins, also a mining engineer of the Bureau. The samples were analyzed by the authors. The table and the comments following touch upon the ventilation in the different mines.

Calculations to show the amount of black-damp present were made only for samples that were not affected by blasting operations in the mines.

TABLE 9.—*Data regarding samples of air collected in certain metal mines.*

MINE 1, TONOPAH, NEV.

Lab. No.	Place of sampling.	Temperature. Wet-bulb.	Temperature. Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp. CO_2	Black damp. N_2	Black damp. CO_2+N_2	Composition of black damp. CO_2	Composition of black damp. N_2	Remarks.
		°F.	°F.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	
4595	West drift, 1,100-foot level	73	84	61	0.05	20.80	79.15						Blasted 3.15 a. m.; sampled 10 a. m.
4596	Face of east drift, 1,200-foot level	87.5	90	91	.11	20.65	79.24						Blasted 3 a. m.; sampled 11.20 a. m.; compressed-air ventilation.
4597	21 raise, off crosscut	87	89	93	.17	20.62	79.21						; crosscut is dead
4598	Shaft, above 900-foot level				.16	20.63	79.21						going up cage.
4599	East drift, No. 7 raise, 1,200-foot level, fan intake.				.07	20.75	79.18						f air per minute
4600½	Top of raise, No. 3 east side, 1,400-foot level.	98	99	97	.25	20.68	79.07						m.
4601	Shaft, above 900-foot level				.07	20.70	79.23						
4602	Face of crosscut 1055	78	86	71	.06	20.65	79.29						
4604	Face of east drift, 400-foot level	90	95	83									
4605	0-foot level (air	82	92	66	.10	20.58	79.32						
4606	vein, off cross-	87	89	93	.04	20.84	79.12						Blasted during night; sampled 11.40 a. m.
4609	Face of west drift, off crosscut, 1,200-foot level.	85	91	79	1.37	18.49	80.14						Blasted 3 a. m.; sampled 1.50 p. m., after place had been mucked out; only ventilation from compressed air.
4610	Face of east drift, on 1,200-foot level	88	95.5	76	.24	20.06	79.70						Sampled 8.15 a. m.; blasted 3 a. m.; man came to work 7 a. m.; end of pipe from fan, 45 feet back from face.
4611	20 feet above sill at top of raise, west drift, shaft vein, off north crosscut, 1,166-foot level.	90	93	89	.31	19.91	79.78						Sampled 10.45 a. m.; blasted 2.30 a. m.; no ventilation but compressed air; one man shoveling.
4612	Intake of fan, raise, east drift, 1,200-foot level.	81	90	69	.12	20.30	79.58						Sampled 8.10 a. m.; blasted during night at 3 a. m.; fan set so as to send the same air to the face repeatedly.

4614	Face, No. 7 crosscut, east drift, 1,300-foot level.	84	89.5	81	.27	20.06	79.67						Sampled 9.40 a. m.; blasted between 12 and 3 a. m.; 30 feet ahead of ventilation; only ventilation from 1-inch compressed-air pipe running one-eighth force, 20 feet back from face.
4615	Top of raise 11, 25 feet above sill; shaft vein, west drift, off northeast crosscut, 1,166-foot level.	91.5	93	95	.66	19.68	79.66						Only ventilation from compressed air.
	MINE 2, KELLOGG, IDAHO.												
4036	Return from stope, near capping station.				1.55	19.44	79.01						Powder smoke rather thick.
4037	....do		81		1.58	19.20	79.22						of it from powder; velocity, cubic feet of air per minute.
4038	Top floor of stope		81										stoping drill about 40 feet
4039	Top of bench raise		64		.61	20.10	79.29						een going 35 minutes; four
4040	Main return, tunnel		56										Velocity, 655; area, 4.8 by 7.1; air, 22,300; sampled 5 p. m.
4041	....do		56		.06	20.84	79.10						Velocity, 643; area, 4.8 by 7.1; air, 21,930; sampled 5.30 p. m.
4042	Return air, tunnel		56		1.41	19.53	79.06						Velocity, 655; area, 4.8 by 7.1, air, 22,300; sampled 4.30 p. m.
4043	....do		56										Velocity, 655; area, 4.8 by 7.1, air, 22,300; part of air came from another mine.
4044	Crosscut, thirteenth level												Sampled at beginning of day shift; ventilation entirely dependent on compressed air.
4045	....do		66		.25	20.73	79.02						Sampled 6.25 p. m.; no compressor running; blasted at 4 p. m.
4050	Thirteenth level, face of first drift right, leaving shaft.		66		.23	20.63	79.14						Sampled 7.20 p. m.; no air running, blasted 4 p. m.; air probably turned on and let run half force from 4 to 7 p. m.
4052	Return from stope, near capping station.				1.49	19.29	79.22						Smoke, apparently from fuse.
4053	Intake of No. 2 shaft		54		.38	20.59	79.03						Velocity, 225[a]; area, 6.5 by 6.2[b]; air, 9,050[c]; 30 to 35 men work in this current before it reaches point of sampling.
4054	Breast of raise from twelfth level		64		.49	20.30	79.21						Small shot fired 12.30 p. m.; sampled 2.20 p. m.; only ventilation from compressed air; not running at time
4055	Face of drift on twelfth level, near floor.		66		1.20	19.70	79.10						m.; round of holes fired 4 p. m.; air one-fourth to one-half force.
4056	Return from stope, near capping station.				1.59	19.30	79.11						p. m.; not as much smoke as at 5 p. m.; may be going down shaft.

[a] Velocity of air current in feet per minute.
[b] Feet.
[c] Volume of air current in cubic feet per minute.

TABLE 9.—*Data regarding samples of air collected in certain metal mines*—Continued.

MINE 2, KELLOGG, IDAHO—Continued.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO_2	O_2	N_2	Black damp.			Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.					CO_2	N_2	CO_2+N_2	CO_2	N_2	
		°F.	°F.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	
4077	Top of raise	......	66	......	0.75	16.43	82.82	......	......	......	......	......	by 5.1; air, 6,000; 60 men work-
4078	Stope near bottom of raise	......	84	......	.30	20.17	79.53	0.27	3.37	3.64	7.4	92.6	per cent; dead end, one man
4079	....do	......	81	......	.13	20.53	79.34	.10	1.82	1.92	5.2	94.8	per cent; dead end, one man

MINE 3, BUTTE, MONT.

Lab. No.	Place of sampling.	Wet-bulb.	Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp CO_2	Black damp N_2	Black damp CO_2+N_2	Composition CO_2	Composition N_2	Remarks.
4099	Return to shaft, south side	84	85	96	.29	20.21	79.50	.26	3.19	3.45	7.5	92.5	Velocity, 260; area, 4.40 by 6.5; air, 7,440.
4098	1675 crosscut, return to shaft	78.5	80	94	.44	20.02	79.54	.41	3.94	4.35	9.4	90.6	Velocity, 490; area, 3 by 6.1; air, 8,950; sampled, 9.45 a. m.; this is main return to shaft from north side
4102	2410 stope, sixth floor	83.5	84.5	96	.29	20.26	79.45	.26	2.95	3.21	8.1	91.9	g.
4103	2801 crosscut, 100 feet north of vein A	89.5	90.5	96	.19	20.52	79.29	.16	1.81	1.97	8.1	91.9	'as from one air drill
4104	2418-C raise, eleventh floor	83	85	92	.31	20.41	79.28	.28	2.21	2.49	11.2	88.8	m.; air running on g; only ventilation,
4105	1601 crosscut, at vein near stable	75	75.5	96	.06	20.84	79.10	.03	.41	.44	6.8	93.2	3,560; part of main
4106	2402-C drift, near face	83	84	96	.16	20.67	79.15	.15	1.10	1.25	12.0	88.0	shot at 2 a. m.; no air turned on till 8.45 a. m., when full head was turned on; sampled, 9.10 a. m., when air felt better near face than back in drift.
4107	Shaft between 400-foot and 800-foot levels.	......	......	......	......	......	......	......	......	......	......	......	Sample taken from cage at 9.30 a. m. before rock-hoisting started.
4108	Return from 2262-C raise, at 2000 level.	76	76.5	98	......	......	......	......	......	......	......	......	
4109	Shaft above 700 level, from cage after passing 700-foot level going to surface.	76	76	100	.10	20.29	79.61	......	......	......	......	......	Sampled at noon.
4110	Face of 2601 drift	84.5	87	90	.09	20.58	79.33	......	......	......	......	......	Sampled at 11.15 a. m.; round blasted in face at 2 a. m.; point of sampling 300 to 400 feet ahead of air; two compressed-air hoses blowing air into or near face.

MINE 4, BUTTE, MONT.

4100	1206 crosscut, 150 feet from 1201 crosscut, at fire wall.	92.5	108.5	60	.42	19.76	79.82	......	......	......	......	......	Velocity, 775; area, 1 foot 8 inches by 4 feet 7 inches; air, 6,300; respiration labored where sample was taken.
4101	Third floor of stope 1809	79	81.5	90	.35	20.26	79.39	.32	2.89	3.21	10.0	90.0	Floor is 3 sets long and 1 set wide; no raise through to next level.
4096	1200 station of No. 1 shaft	84	88	85	.23	20.40	79.37	.20	2.34	2.54	7.9	92.1	
4097	1260 station near junction 1284, return to air shaft.	91.5	96	84.5	.27	20.10	79.63	.24	3.73	3.97	6.0	94.0	9 inches; room on
4282	First floor of 1491 stope	87.5	88.5	98	.18	20.19	79.63	.15	3.39	3.54	4.2	95.8	
4283	Twenty-second floor of stope 1637	86	87	96	.07	20.85	79.08	.04	.35	.39	10.2	89.8	irculation
4285	Top of fourth floor of 1492 raise	85	87.5	92	.19	20.23	79.58	.16	3.19	3.35	4.8	95.2	
4286	Sixth floor of stope 1477	88	89	96	.15	20.47	79.38	.12	2.08	2.20	5.5	94.5	mple was
4224	1414, 10 feet north of stope 1450	84	85	96	.11	20.41	79.48	.08	2.41	2.49	3.2	96.8	e air goes
4225	On 1409 drift at 1453 chute	82.5	83.5	96	.15	20.36	79.49	.12	2.61	2.73	4.4	95.6	returning
4227	Second floor of stope 1678	81	81.25	99	.60	19.73	79.67	.57	5.17	5.74	9.9	90.1	is holed
4229	1900 level, about 86 feet from No. 1 shaft	85.5	87	94	.09	20.86	79.05	.06	.28	.34	17.6	82.4	running es before
4230	Breast of 2007 drift	89.5	90	98	.17	20.55	79.28	.14	1.68	1.82	7.7	92.3	
4231	Outlet of main fan, at surface	87	89	92	.22	20.41	79.37	.19	2.30	2.49	7.6	92.4	
4232	Eighteenth floor of 1641 stope	87	88	......	.13	20.48	79.39	.10	2.07	2.17	4.6	95.4	One man working 30 feet from raise which was holed through.

MINE 5, BUTTE, MONT.

4141	Second floor of stope 2390	78	80	92	0.07	20.92	79.01	......	......	......	......	......	third floor near where
4142	Sill floor of stope 2326 (survey point 92).	85.5	86	99	.14	20.73	79.13	......	......	......	......	......	ing, owing to recent
4143	Face of 2880 drift, 125 feet west of 2886	90	91	96	.17	20.79	79.04	......	......	......	......	......	, discharging air into one-half force; sample blasted 2 a. m.; sam-
4144	Station of 3000 level	81	82	96	.16	20.71	79.13	......	......	......	......	......	only; blasted 8 a. m.;
4145	Fan conduit	75	75.5	98	.21	20.57	79.22	0.18	1.55	1.73	10.4	89.6	, 33,650.
4146	2017 raise on 1900	86	86	100	.35	20.36	79.29	.32	2.41	2.73	11.7	88.3	, 9,800.
4147	1900 water crosscut, west of 2017	85	88	80	.33	20.42	79.25	.30	2.14	2.44	12.3	87.7	Face is in old worked out and filled stope; compressed air turned on about one-fourth force, 10 feet back from face.
4148	Breast of stope, twelfth floor, 10 sets east of 2270 raise.	70	72.5	89	.07	20.87	79.06	.04	.25	.29	13.8	86.2	

TABLE 9.—*Data regarding samples of air collected in certain metal mines*—Continued.

MINE 2, KELLOGG, IDAHO—Continued.

Lab. No.	Place of sampling.	Temperature. Wet-bulb.	Temperature. Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp. CO_2	Black damp. N_2	Black damp. CO_2+N_2	Composition of black damp. CO_2	Composition of black damp. N_2	Remarks.
		°F.	°F.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	
4077	Top of raise	......	66	......	0.75	16.43	82.82	......	......	......	......	......	ırea, 3.8 by 5.1; air, 6,000; 60 men work-
4078	Stope near bottom of raise	......	84	......	.30	20.17	79.53	0.27	3.37	3.64	7.4	92.6	ırly 100 per cent; dead end, one man is place.
4079	....do	......	81	......	.13	20.53	79.34	.10	1.82	1.92	5.2	94.8	Humidity, nearly 100 per cent; dead end, one man working in this place.

MINE 3, BUTTE, MONT.

Lab. No.	Place of sampling.	Temperature. Wet-bulb.	Temperature. Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp. CO_2	Black damp. N_2	Black damp. CO_2+N_2	Composition of black damp. CO_2	Composition of black damp. N_2	Remarks.
4090	Return to shaft, south side	84	85	96	.29	20.21	79.50	.26	3.19	3.45	7.5	92.5	Velocity, 260; area, 4.40 by 6.5; air, 7,440.
4098	1675 crosscut, return to shaft	78.5	80	94	.44	20.02	79.54	.41	3.94	4.35	9.4	90.6	Velocity, 490; area, 3 by 6.1; air, 8,960; sampled, 9.45 a. m.; this is main return to shaft from north side on 1,600 level.
4102	2410 stope, sixth floor	83.5	84.5	96	.29	20.26	79.45	.26	2.95	3.21	8.1	91.9	
4103	2801 crosscut, 100 feet north of vein A	89.5	90.5	96	.19	20.52	79.29	.16	1.81	1.97	8.1	91.9	from one air drill
4104	2418-C raise, eleventh floor	83	85	92	.31	20.41	79.28	.28	2.21	2.49	11.2	88.8	; air running on only ventilation,
4105	1501 crosscut, at vein near stable	75	75.5	96	.06	20.84	79.10	.03	.41	.44	6.8	93.2	,560; part of main
4106	2402-C drift, near face	83	84	96	.18	20.67	79.15	.15	1.10	1.25	12.0	88.0	ıot at 2 a. m.; no ll head was turned elt better near face
4107	Shaft between 400-foot and 800-foot levels.	......	......	......	......	......	......	......	......	......	......	......	. m. before rock-
4108	Return from 2252-C raise, at 2000 level.	76	76.5	98	......	......	......	......	......	......	......	......	
4109	Shaft above 700 level, from cage after passing 700-foot level going to surface.	75	76	100	.10	20.20	79.61	......	......	......	......	......	Sampled at noon.
4110	Face of 2601 drift	84.5	87	90	.09	20.58	79.33	......	......	......	......	......	Sampled at 11.15 a. m.; round blasted in face at 2 a. m.; point of sampling 300 to 400 feet ahead of air; two compressed-air hoses blowing air into or near face.

MINE 4, BUTTE, MONT.

4100	1206 crosscut, 150 feet from 1201 crosscut, at fire wall.	92.5	106.5	60	.42	19.76	79.82	......	......	......	......	......	Velocity, 775; area, 1 foot 8 inches by 4 feet 7 inches; air, 6,300; respiration labored where sample was taken.
4101	Third floor of stope 1869	79	81.5	90	.35	20.26	79.39	.32	2.89	3.21	10.0	90.0	Floor is 3 sets long and 1 set wide; no raise through to next level.
4096	1200 station of No. 1 shaft	84	88	85	.23	20.40	79.37	.20	2.34	2.54	7.9	92.1	
4097	1260 station near junction 1264, return to air shaft.	91.5	96	84.5	.27	20.10	79.63	.24	3.73	3.97	6.0	94.0	.9 inches; room on
4282	First floor of 1491 stope	87.5	88.5	96	.18	20.19	79.63	.15	3.39	3.54	4.2	95.8	
4283	Twenty-second floor of stope 1637	86	87	96	.07	20.85	79.08	.04	.35	.39	10.2	89.8	rculation marked.
4285	Top of fourth floor of 1492 raise	85	87.5	92	.19	20.23	79.58	.16	3.19	3.35	4.8	95.2	Ventilation, compressed air only.
4286	Sixth floor of stope 1477	88	89	96	.15	20.47	79.38	.12	2.08	2.20	5.5	94.5	Two repair men had just left place when sample was taken.
4224	1414, 10 feet north of stope 1450	84	85	96	.11	20.41	79.48	.08	2.41	2.49	3.2	96.8	1414 is air course on 1400; sample taken where air goes into shaft to return to surface.
4225	On 1409 drift at 1453 chute	82.5	83.5	96	.15	20.36	79.49	.12	2.61	2.73	4.4	95.6	Velocity, 190; area, 3.8 by 5.7, air, 4,080; air returning to No. 1 shaft.
4237	Second floor of stope 1678	81	81.25	99	.60	19.73	79.67	.57	5.17	5.74	9.9	90.1	ear raise which is holed
4229	1900 level, about 86 feet from No. 1 shaft	85.5	87	94	.09	20.86	79.05	.06	.28	.34	17.6	82.4	air, which was running ntil two minutes before
4230	Breast of 2007 drift	89.5	90	98	.17	20.55	79.28	.14	1.68	1.82	7.7	92.3	
4231	Outlet of main fan, at surface	87	89	92	.22	20.41	79.37	.19	2.30	2.49	7.6	92.4	
4232	Eighteenth floor of 1641 stope	87	88	.....	.13	20.48	79.39	.10	2.07	2.17	4.6	95.4	One man working 20 feet from raise which was holed through.

MINE 5, BUTTE, MONT.

4141	Second floor of stope 2390	78	80	92	0.07	20.92	79.01	......	......	......	......	......	ir where
4142	Sill floor of stope 2326 (survey point 92)	85.5	86	99	.14	20.73	79.13	......	......	......	......	......	recent
4143	Face of 2880 drift, 125 feet west of 2886	90	91	96	.17	20.79	79.04	......	......	......	......	......	air into sample n.; sam-
4144	Station of 3000 level	81	82	96	.16	20.71	79.13	......	......	......	......	......	8 a. m.;
4145	Fan conduit	75	75.5	98	.21	20.57	79.22	0.18	1.55	1.73	10.4	89.6	
4146	2017 raise on 1900	86	86	100	.35	20.36	79.29	.32	2.41	2.73	11.7	88.3	
4147	1900 water crosscut, west of 2017	85	88	99	.33	20.42	79.25	.30	2.14	2.44	12.3	87.7	npressed air turned on about one-fourth force, 10 feet back from face.
4148	Breast of stope, twelfth floor, 10 sets east of 2270 raise.	70	72.5	89	.07	20.87	79.06	.04	.25	.29	13.8	86.2	

TABLE 9.—*Data regarding samples of air collected in certain metal mines*—Continued.

MINE 5, BUTTE, MONT.—Continued.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO_2	O_2	N_2	Black damp.			Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.					CO_2	N_2	CO_2+N_2	CO_2	N_2	
		°F.	°F.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	
4149	Breast of stope, 2 sets east of 2809 raise.	90	91	96	0.12	20.72	79.16	.09	.92	1.01	8.9	91.1	Sampled 2.35 p. m.; blasted on previous shift, 2 a. m.; compressed air running one-third force; no drilling going on.
4152	Face of drift 2825	90	91	96	.13	20.69	79.18	.10	1.05	1.15	8.7	91.3	1 minute ,720 cubic ce; place
4153	Raise short distance below fan	78	78	100	.26	20.36	79.38	.23	2.50	2.73	8.4	91.6	e.
4154	2444 on 2200 level	82.5	86	87	.17	20.73	79.10	.14	.82	.96	14.6	85.4	
4156	Top of 2370 stope	85	86.5	94	.16	20.72	79.12	.13	.88	1.01	12.9	87.1	Av. 2.515
4157	Top of 2401 raise on 2400 level	83	85	92	.16	20.65	79.19	.13	1.22	1.35	9.6	90.4	
4158	Top of 2835 raise	87.5	89	94	.18	20.64	79.18	.15	1.24	1.39	10.8	89.2	
4159	Top of raise 2209	74	77	87	.08	20.83	79.09	.05	.34	.39	12.9	87.1	
4161	Top of main air raise on 1100 level				.30	20.35	79.35	.27	2.51	2.78	9.7	90.3	
4162	Air raise just below 1000 level	80	80	100	.21	20.54	79.25	.18	1.69	1.87	9.6	90.4	
4163	1000 level, at door near raise	80.5	81	99	.22	20.55	79.23	.19	1.63	1.82	10.4	89.6	
4164	Incline near surface	78	91	57	.13	20.62	79.25	.10	1.39	1.49	6.7	93.3	
4165	1900 water crosscut, west of 2017				.31	20.49	79.20	.28	1.83	2.11	13.3	86.7	

	MINE 6, IRONWOOD, MICH.											
4315	No. 9 crosscut, 24 level, 225 feet north of rock drift.	60	61	....	0.14	20.76	79.10	0.11	0.71	0.82	13.4	86.6	Velocity, 450; area, 54. air, 24,300; air passing timbers.
4316	K shaft, 50 feet above 19th level.......	55	57.5	....	.11	20.81	79.08	.08	.50	.58	13.8	86.2	Air, 35,800; air passing timbers.
4317	.....do..................................	55	57.5	....	.06	20.81	79.13	.03	.55	.58	5.2	94.8	Do.
4318	Drift, 347 feet west of 168 raise........	61	62	....	.10	20.72	79.18	.07	.94	1.01	......	......	Velocity, 360; area, 42.25; air, 15,210; air passing timbers.
4319	.....do..................................	61	62	....	.10	20.77	79.13	.07	.70	.77	9.1	90.9	Velocity, 360; area, 42.25; air, 15,210; air passing timbers.
4320	888 feet west of K shaft, 24 level.......	59	60	....	.10	20.84	79.06	.07	.37	.44	15.9	84.1	Velocity, 600; area, 60; air, 36,000; air passing timbers.
4321	.....do..................................	59	60	....	.07	20.78	79.15	.04	.68	.72	5.6	94.4	Do.
4322	Drift, 592 feet west of 168 raise........	61.5	62.5	....	.09	20.76	79.15	.06	.76	.82	7.3	92.7	Velocity, 265; area, 40.3; air, 10,680; air passing timbers.
4323	.....do..................................	61.5	62.5	....	.10	20.77	79.13	.07	.70	.77	9.1	90.9	Do.
4324	No. 9 crosscut, 24 level, 225 feet north of rock drift.	60	61	....	.07	20.81	79.12	.04	.54	.58	6.9	93.1	Velocity, 450; area, 54; air, 24,300; air passing timbers.
	MINE 7, TROJAN, S. DAK.												
4325	Breast of C left stope..................	48	49	94	0.26	20.37	79.37	......	......	......	......	......	
4326	Breast of D stope.......................	49	51	88	.38	20.17	79.45	......	......	......	......	......	
4327	Breast of C stope.......................	46	47	93	.56	19.88	79.56	......	......	......	......	......	oke in air at
4328	Raise....................................	50	51	94	.46	19.95	79.59	......	......	......	......	......	1.40 p. m.
4329	Breast of M stope.......................	48	48.5	97	.20	20.31	79.49	......	......	......	......	......	m.
4343	West crosscut...........................	42	43.5	90	......	......	......	......	......	......	......	......	
4344	Tunnel..................................	42	44	86	.08	20.81	79.11	......	......	......	......	......	one shoveling, one running Leyner place on previous day shift; sam
	MINE 8, TROJAN, S. DAK.												
4345	Crosscut raise..........................	44	46	87	0.12	20.73	79.15	......	......	......	......	......	Blasted 18 hours before sample was taken; sampled, 10.20 a. m.
	MINE 9, TONOPAH, NEV.												
4617	Toilet in end of crosscut, south of drift 515, shaft.	......	......	......	0.04	20.88	79.08	......	......	......	......	......	

TABLE 9.—*Data regarding samples of air collected in certain metal mines*—Continued.

MINE 10, TONOPAH, NEV.

Lab. No.	Place of sampling.	Temperature.		Relative humidity.	CO_2	O_2	N_2	Black damp.			Composition of black damp.		Remarks.
		Wet-bulb.	Dry-bulb.					CO_2	N_2	CO_2+N_2	CO_2	N_2	
		°F.	°F.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	
4624	Face of 1003 drift west	75	80	80	0.09	20.63	79.28						...s.
4625	Face of 1018 south crosscut, 30 feet from 1003 west drift.	77	80	88	.14	20.74	79.12						...pled, 8.45 a. m.; fan shut off ... m.; fan delivering 855 cubic ...t 100 feet back from face. ...inute from fan pipe must have
4627	1,000-foot level station	77	78	96	.13	20.70	79.17	0.10	1.01	1.11	9.0	91.0	

MINE 11, LEAD, S. DAK.

Lab. No.	Place of sampling.	Wet-bulb.	Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp CO_2	Black damp N_2	Black damp CO_2+N_2	Composition CO_2	Composition N_2	Remarks.
4421	Face of No. 12 west crosscut on 1,700-foot level.	61	62.5	92	0.12	20.71	79.17	0.09	0.97	1.06	8.5	91.5	Dead end; only ventilation is from compressed air.
4422	Footwall west of mule chute on 200-foot level.	78.5	80	94	.38	20.43	79.19	.35	2.05	2.40			
4423	Hanging-wall header east of 13,200-foot level.	66	66.5	98	.20	20.68	79.12	.17	1.03	1.20	14.2	85.8	Return air; most of it from No. 4 and No. 6 districts of 300-foot level; perceptible current, but not enough to allow velocity to be measured.
4424	Sill floor, 16 stope south on 1,700-foot level.	61.5	62.5	96	.22	20.74	79.04						Considerable powder smoke.
4425	Sill floor 4 west, vertical stope, 400-foot level.	79	82	88	.31	20.21	79.48						Considerable powder smoke; good current of air; powder smoke lifted quickly and air was clear short time after sample was taken.
4426	...orthwest stope,	71	73	91	.24	20.40	79.36	.21	2.33	2.54	8.2	91.8	Raise caved and blocked; not much air circulating.
4427	...t level	78	80	92	.30	20.26	79.44	.27	2.94	3.21	8.4	91.6	Drawing ore from chute; no air circulating.
4428	...orthwest stope,	88	89	89	.23	20.28	79.49	.20	2.91	3.11	6.5	93.5	Only ventilation from compressed air; timbers badly crushed; odor of decayed timber.
4429	...thwest stope of 300-foot level.	67	67	100	.25	20.38	79.37						Air foggy.
4431	No. 5 stope on 1,000-foot level	59	60	95	.10	20.46	79.44						Some powder smoke noticeable in air.
4432	No. 5 stope, 500-foot level, on hanging-wall side where waste was being dumped.	55.5	56.5	94	.17	20.57	79.26						

4433	No. 3 south on 1,250-foot level	58	58.5	97	.02	20.69	79.29						Not taken in dead end; considerable air was moving through drift at time of sampling, but it contained some powder smoke from block holing.
4434	Drift in No. 3 north of vertical stope, 400-foot level.	78.5	82	86	.34	20.47	79.19	.31	1.90	2.21	14.0	86.0	Ground caved.
4435	1,000-foot level at top raise from No. 6 stope, east 1100.	58	59	95	.13	20.39	79.48						Considerable powder smoke from block holing.
4436	Eighth floor, No. 9 stope, 1,250-foot level.	59.5	60.5	95	.19	20.50	79.31	.16	1.91	2.07	7.7	92.3	Two men working with drill on top of broken rock at 8.30 a.
4437	Sill floor, stope 2, 500-foot level	59	60	95	.10	20.61	79.29						Twelve sampl turned cleared
4438	Second floor, No. 4 west vertical stope, 400-foot level.	100.5	101.5	97	1.11	19.24	79.65	1.08	7.00	8.08	13.4	86.6	Beads of moisture hanging on timbers; sampled 11.40 a. m.
4439	Ninth floor, No. 3 north center stope, 600-foot level.	75	79	84	.10	20.61	79.29						crushed; dead end; odor of decaying
4441	Shaft header, 600 feet from face, 1,700-foot level.	61	62	95	.20	20.59	79.21						shaft header where powder smoke was
4442	Fifth floor, No. 2 north stope, 600-foot level.	75	79	84	.20	20.43	79.37	.17	2.23	2.40	7.1	92.9	; odor of decaying timber.
4443	Fifth floor, No. 2 west stope, 600-foot level.	76	78	92	.19	20.44	79.37	.16	2.19	2.35	6.8	93.2	Two had been drilling shortly before sample was taken.

MINE 12, VETERAN, NEV.

4458	Small raise (No. 3) near breast of No. 59 drift.	85	88	89	0.47	19.88	79.65	0.44	4.58	5.02	8.8	91.2	Sampled after raise had been connected with sublevel.
4459	Inby toilet boxes	64	65	96	.43	20.11	79.46	.40	3.52	3.92	10.2	89.8	
4460	Bottom 40 winze where drift was being widened.	64	66.5	89	.61	20.16	79.23	.58	3.11	3.69	15.7	84.3	ad been made for venti-
4461	Return air at main shaft	64	67	86	.69	19.93	79.38	.66	4.12	4.78	13.8	86.2	but velocity not great measurement.
4462	Face 34 winze incline	73	76	78	1.04	19.37	79.59						air pipe connected to fan of air per minute 25 feet back from face.
4463	Near bottom of drill hole for suction fan	67	68	95	1.10	18.77	80.13	1.07	9.25	10.32	10.4	89.6	Velocity, 702; area, 1.9 by 2.15; air, 2,870.
4464	In 85 sublevel just above 59 (No. 3 raise).				1.22	18.34	80.44						Blasted, 11.25 a. m.
4465	Face of No. 37 drift (retreating)	78	81	88	.32	20.38	79.30						blowing one-
4466	Small raise (No. 3) near breast of 59 drift.	93	98	83	.71	19.30	79.99	.68	7.11	7.79	8.7	91.3	candle extin- sampled 8.55
4467	Face of sublevel 55				.84	19.54	79.62						
4468	Crosscut 4	76	78	92	.57	20.06	79.37	.54	3.62	4.10	13.0	87.0	ut 15 minutes n compressed air just before

TABLE 9.—*Data regarding samples of air collected in certain metal mines*—Continued.

MINE 12, VETERAN, NEV.—Continued.

Lab. No.	Place of samp			Relative humidity.									
		°F.	°F.	P.ct.	P.ct.	P.ct.							
4469	Mule barn	73	77	83	0.83	19.77	7						
4470	West drift (30 raise)	69	71	91	.76	19.81	7						
4471	Face of No. 85 sublevel, east end	83	87	....	.62	19.73	7						
4472	Near face of No. 85 sublevel), west end	85	88	89	.60	19.77	7						
4473	South end of 17 winze	70.5	71.5	95	1.17	18.85	7						
4474	Sixth crosscut through pillar	76	80	84	.79	19.75	7						
4475	East end of No. 73 sublevel	74	77.5	94	.30	20.40	7						
4476	East end of sublevel 73, near raise 55	73	75	91	.67	20.92	7						
4477	Near face of No. 59 drift (retreating)	84	89	82	.61	19.82	7						
4478	Face of sublevel 65, west side	70	72	91	.92	19.05	80.03	......	......	......	......	......	
4479	Sublevel 65, east side	67	69	91	.94	19.11	79.95	......	......	......	......	......	
4540	Second sublevel above 13th level, between chutes 50 and 51.	78	80	....	(1)	......	......	......	......	......	......	......	
4541	...do	79	80	....	(1)	......	......	......	......	......	......	......	Do.
4542	Face east cave, over 45 chute, 12th level.	76	79	....	.73	20.17	79.10	......	......	......	......	......	Before blasting.
4543	...do	76	70	....	.71	20.14	79.15	......	......	......	......	......	Do.

4544	Mouth of cave in 45 chute, 12th level (1½ hours after blast).	75	75	......	(1)	......	......	......	......	......		......	Some powder smoke.
4545	......do	75	75	......	(1)	......	......	......	......	......		......	Do.
4548	50 feet below collar B shaft (upcast)	61	61	......	.26	20.08	79.06	......	......	......		......	
4549	......do	61	61	......	.28	20.68	79.04	......	......	......		......	

MINE 13, TONOPAH NEV.

4799	Face 409 south crosscut	68	71	81	.15	20.70	79.15	......	......	......	......	......	
4800	3 west drift, off 3 south crosscut, 500-foot level.	65	70	78	.04	20.91	79.05	......	......	......	......	......	
4801	Top of 610 raise	71	73	91	.42	20.44	79.14	.39	1.96	2.35	16.5	83.5	
4802	605 east drift	63	66	86	.06	20.79	79.15	.03	.05	.68	4.4	95.6	
4803	No. 1 stope, 500-foot level	68.5	69.5	95	.10	20.73	79.17	......	......	......	......	......	
4804	Compressed air from face, 409 south crosscut.	......	......	......	.06	20.88	79.06	.03	.23	.25	12.0	88.0	Sample of compressed air taken 7.26 p. m.
4805	No. 1 shaft above 400-foot level	......	......	......	.08	20.80	79.12	......	......	......	......	......	Return air from No. 1 shaft; sampled 7.10 p. m.; blasted 3.15 p. m.
4806	Face 606, intermediate level	64	69	78	.13	20.69	79.18	......	......	......	......	......	) smoke visible; blasted
4807	602 east heading	61	79	34	.13	20.72	79.15	......	......	......	......	......	p. m.
4808	Face No. 3 south crosscut, 600-foot level	65	70	78	.06	20.80	79.14	......	......	......	......	......	pile at 9 p. m.; blasted
4809	610 raise	69	73	83	.20	20.69	79.11	......	......	......	......	......	mpled 9.35 p. m.

MINE 14, TONOPAH, NEV.

4810	422 east drift (face)	70	73	87	.18	20.64	79.18	......	......	......	......	......	
4811	No. 332 crosscut	71	73	91	.22	20.62	79.16	......	......	......	......	......	
4812	No. 24 raise, 565-foot intermediate level.	74	77	87	.52	20.32	79.16	......	......	......	......	......	was taken. Blasted 3.30 p. m.; sampled at 10.15 a. m.; one man running Waugh air drill; ventilation connection not very good.

[1] No sample.

TABLE 9.—*Data regarding samples of air collected in certain metal mines*—Continued.

MINE 14, TONOPAH, NEV.—Continued.

Lab. No.	Place of sampling.	Temperature. Wet-bulb.	Temperature. Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp. CO_2	Black damp. N_2	Black damp. CO_2+N_2	Composition of black damp. CO_2	Composition of black damp. N_2	Remarks.
		° *F.*	° *F.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
4813	No. 24 east drift, 565-foot intermediate level.	72	73	91	.30	20.54	79.16	.27	1.60	1.87	14.5	85.5	
4814	Top raise, 465-foot level	69	70	95	.04	20.91	79.05						3.50 a. m.
4815	Air receiver on 615-foot level				.05	20.93	79.02						receiver; nine.
4816	20 intermediate stope, 565-foot level	74	77.5	86	.36	20.39	79.25						drilling, 44 cubic
4817	West end of vein near face, crosscut 628, cutting stope.	62	65	86	.14	20.61	79.25	.11	1.43	1.54	7.1	92.9	working

MINE 15, TONOPAH, NEV.

Lab. No.	Place of sampling.	Temperature. Wet-bulb.	Temperature. Dry-bulb.	Relative humidity.	CO_2	O_2	N_2	Black damp. CO_2	Black damp. N_2	Black damp. CO_2+N_2	Composition of black damp. CO_2	Composition of black damp. N_2	Remarks.
4768	600-C stope	58	71½	48	0.06	20.36	79.58						Blasted 2.20 a. m.; sampled 8.30 a. m.; stope connected, not a dead end.
4769	501 hangingwall crosscut	46	58½	41	.07	20.35	79.58						
4770	Intake of fan on 600-foot level				.09	20.81	79.10						
4771	Top raise 662	76	79½	88	.23	20.36	79.31						
4772	602 drift, about 20 feet ahead of air pipe.	74	78½	84	.10	20.07	79.23	.07	1.18	1.25	5.6	94.4	
4773	In main shaft, above the 400-foot level				.08	20.73	79.19	.06	.91	9.6	5.2	9	
4774	601 winze	68	72.5	61	.07	20.78	79.14						down a winze; only ventilation from compressed air.
4775	Raise 548 west	71	71	100	.71	17.77	81.53	.78	14.43	15.10	4.5	95.5	Candle was extinguished where sample was taken; may have a little gas from strata.

4776	70 feet from top of 641 raise	73	74	96	.29	20.56	79.15	......	......	......	......	......	Two raises come together here; no connection through for ventilation; dead end; blasted 3.20 p. m.; sampled 9.30 a. m.
4777	600 east stope	74	75	96	.14	20.74	79.12	......	......	......	......	......	Dead end; blasted in raise nearby, 2.20 a. m; blasted here at 3.20 a. m.; sample taken at 9.10 a. m.
4778	Compressed-air sample	......	......	......	.06	20.91	79.03	......	......	......	......	......	Sample taken on the surface about 3.50 p. m.
4779	Top of 669 raise	76	78	92	.16	20.58	79.26	.13	1.55	1.68	7.7	92.3	Dead end; man had not worked here for a week; only ventilation from compressed air; man just starting to work when sample was taken.

COMMENTS ON RESULTS OF ANALYSIS.

Only one of the samples (laboratory No. 4609) of air from Mine 1 contained a proportion of oxygen that was either low or excessive as compared with the oxygen content of ordinary air. All of the samples were collected several hours after blasting operations. Hence the mine air was not vitiated in any appreciable degree at the time the samples were collected. All of the temperatures observed at the time of sampling were high—too high to allow men to do good work.

Of the samples collected in Mine 2, four (laboratory Nos. 4036, 4037, 4052, and 4056) contained excessive amounts of carbon dioxide. These samples were all collected at the same place, a few minutes apart. The atmosphere contained powder smoke as the result of a previous blast, hence the atmosphere was vitiated by products of combustion from the blast. Wet-bulb temperatures were not noted in this mine. The dry-bulb temperatures were good except that observed at the time of taking the sample designated laboratory No. 4079. The wet-bulb temperature was not determined in connection with the collection of this latter sample, but the sense of discomfort experienced by the sampler and the moist condition of the place of sampling indicated that the air was almost saturated with moisture. Sample 4079 was the only one, not contaminated with products of combustion from a blast, that had a high percentage of carbon dioxide. The analyses of two of the samples collected in Mine 2 were recalculated to show the percentage of black damp present.

Wet-bulb temperatures observed in Mine 3 exceeded 75° F. The chemical analyses disclosed no low oxygen or high carbon dioxide content. The composition of the black damp ranged from 5.2 to 12 per cent carbon dioxide and from 88 to 94.8 per cent oxygen.

The temperatures and the humidity readings observed in Mine 4 were excessively high. The chemical analyses of the air disclosed no bad condition. Black damp contained 4.2 to 17.6 per cent carbon dioxide and 82.4 to 95.8 per cent oxygen.

The temperatures were higher in Mine 5 than is compatible with the best working efficiency. The chemical analyses disclosed no high carbon dioxide or low oxygen percentages. The black damp contained 6.7 to 13.8 per cent carbon dioxide and 86.2 to 93.3 per cent nitrogen.

Favorable temperatures were found in Mines 6, 7, 8, and 9. The chemical analyses of the air also showed good conditions.

Temperatures in Mine 10 were not so good.

Most of the observed temperatures in Mine 11 were good. There was one noteworthy exception, that noted at the time of taking sample 4438. The carbon dioxide content was also high in that sample.

Considerable powder smoke was present at some places where samples were collected, but no carbon monoxide that might have remained in the air after the shooting was found.

Many of the temperatures observed in Mine 12 were low. Some exceeded 75° F., wet bulb. The carbon dioxide and oxygen contents in some of the samples were low, especially in samples 4463, 4464, and 4473. Just how much blasting operations affected these samples is problematical. One sample (No. 4464), collected 11 hours after blasting, contained 1.22 per cent carbon dioxide and 18.34 per cent oxygen. In general the chemical analyses of the samples from Mine 12 disclosed higher carbon dioxide and lower oxygen contents than in the samples from any of the other mines.

The temperatures in Mine 12 were favorable.

The temperatures observed in Mines 14 and 15 were good. Only in Mine 12 were samples collected that contained excessively high percentages of carbon dioxide or excessively low percentages of oxygen.

In summing up conditions in these metal mines it can be said that, as regards the chemical analyses of the samples collected, the air, on the whole, was very good. Samples collected several hours after blasting, and when men were mucking out their places, did not contain carbon monoxide from the blasting operations. Changes in the air of metal mines due to oxidation are not as rapid as in coal mines, in spite of the fact that immense volumes of air sweep through coal mines as compared to metal mines, where the ventilation is natural or where air is supplied from compressed-air lines.

The chief trouble lies in the high temperatures, both wet-bulb and dry-bulb; also, the stagnation of the air encountered in many places is so bad as to be detrimental to the health and to the efficiency of the workmen.

OBSERVATIONS REGARDING BLACK DAMP IN CERTAIN METAL MINES.

The average percentage of black damp in certain mines and the average composition of the black damp are shown in the following table:

Average percentage and average composition of black damp in certain metal mines.

Mine No.	Average per cent of black damp.	Composition of black damp.	
		CO_2.	N_2.
		Per cent.	*Per cent.*
2 and 3	2.52	8.4	91.6
4	2.64	7.4	92.6
5	1.60	10.9	89.1
6	.71	7.9	92.1
11	2.78	9.5	90.5
12	5.76	11.8	88.2

The average of all analyses in the above table is 2.67 per cent black damp with a composition corresponding to 9.3 per cent carbon dioxide and 90.7 per cent nitrogen.

The average composition of black damp in metal mines is about the same as that found in coal mines. A difference in its occurrence in the two kinds of mines lies in the amount that can be produced in a given time.

If coal mines depended only on natural ventilation or on compressed air from pipes at working places, the amount of black damp produced would usually exceed far more than it does that produced in metal mines, for coal reacts with oxygen much more rapidly than do most of the rocks, wood, etc., found in metal mines.

SUMMARY.

The most satisfactory definition of the term "black damp" is an accumulation of carbon dioxide and nitrogen in excess of the percentage found in pure atmospheric air.

The principal factors that affect the changes in mine air are (1) the velocity with which the mine air traverses the mine passages; (2) the amount of coal with which it comes in contact; (3) the gaseous (methane) nature of the seam; (4) the nature of the coal as regards its power to react with oxygen; (5) the temperature and the wetness of the mine.

Carbon dioxide must be present in large proportions before it threatens life. A proportion of 3 to 4 per cent of carbon dioxide in air affects the breathing of most people. Men may, however, work for a long time in such an atmosphere, although their efficiency as workmen will be greatly affected and they will become fatigued quickly. The presence in air of as little as 1 or 2 per cent of carbon dioxide is not so much a matter of safety and comfort to those who breathe it as it is of their efficiency as workmen.

Distress is caused in some people when the oxygen content falls to less than 13 per cent. Rapid breathing is produced much more quickly by an excess of carbon dioxide than by a corresponding deficiency of oxygen. The important point to remember is that rapid breathing caused by carbon dioxide starts long before there is any serious danger, whereas rapid breathing caused by a deficiency of oxygen is a grave symptom and points urgently to serious danger.

By acclimatization people live the year round at high altitudes where the air has an oxygen content, by weight, that is the same as that of an atmosphere at sea level containing 12 per cent oxygen by volume. People unaccustomed to such atmospheres, if suddenly plunged into them, experience severe distress. In an experiment conducted by the authors a man lost consciousness temporarily when the oxygen content of an atmosphere he breathed fell to 7 per cent.

Mice and canaries are about as resistive to low-oxygen atmospheres as men; hence they can not be used by exploring parties to give warning of atmospheres that are dangerously low in oxygen.

An excess of oxygen or a diminution of oxygen such as often occurs in many mines, if present in buildings above ground where people congregate, would be indicative of extremely bad ventilation. However, in mines having such an excess or diminution the ventilation may be excellent. In most buildings above ground, the problem of good ventilation is not to correct a diminished content of oxygen or an increased content of carbon dioxide, but to maintain proper temperature and relative humidity, and to keep the air moving, although the presence of an excessive proportion of carbon dioxide, more than 0.1 or 0.2 per cent, may be a reliable indication of air that will produce injurious effects on men. In these buildings the carbon dioxide comes chiefly from the air exhaled by the persons present, and, if fresh air is not admitted, invariably accompanies bad conditions such as stagnant and oppressive air, high temperature, and frequently high humidity. In coal mines the carbon dioxide is principally from the action of the oxygen of the air on the coal, and 0.2 per cent is frequently found in the cool, swiftly moving air of returns where 50,000 or more cubic feet of air is passing per minute. Hence, a proportion that accompanies good conditions of ventilation in a coal mine may indicate extremely poor conditions of ventilation in the room of a house.

A similar statement applies to oxygen. The oxygen content is scarcely ever normal in a coal mine, owing to the gas being absorbed by the coal; in fact, a diminution of 1 per cent is not uncommon. Such a diminution in a building filled with people would be accompanied by intolerable conditions of ventilation. However, even in coal mines the oxygen content of the air should not be allowed to become too low, and the authors believe that it should not fall below 19 per cent. That this limit can easily be maintained in coal mines is indicated by many mine-air analyses made by the authors. The maximum percentage of carbon dioxide allowed in English coal mines is 1.25 per cent.

The principal cause of the depletion of oxygen in coal-mine air and the increase of carbon dioxide is the reaction between the oxygen of the air and coal. Some of the oxygen is actually held dissolved in the coal substance. Part of the oxygen is converted into water, part into carbon dioxide, and part (by far the larger part) is retained as combined oxygen to give compounds richer in oxygen than the coal itself. Part of the carbon dioxide is retained by the coal.

Explosive proportions of methane in air become nonexplosive when the proportion of oxygen in the atmosphere falls below about 14 per cent. Carbon dioxide has only a slightly greater effect in reducing

the explosibility of methane-air mixtures than nitrogen has; for instance, when the oxygen is kept constant at 20 per cent, part of the nitrogen must be replaced by 10 per cent of carbon dioxide to raise the low limit for methane from 5.8 to 6.2 per cent.

The specific gravity of black damp varies considerably in certain mixtures. When methane is present the combined gases may be lighter than air. Great caution should be observed when one detects an accumulation of black damp of lighter density than air, especially in coal mines in which naked lights are used, as this lesser density is probably due to the presence of methane.

Except directly over a fire area or close to a mine fire, a large proportion of carbon dioxide (more than 3 to 5 per cent) is unusual in the air of a coal mine.

An oil-fed flame becomes extinguished when the oxygen in air falls to about 17 per cent; an acetylene flame is extinguished when the oxygen falls to about 12 or 13 per cent.

Lack of oxygen is the important factor in extinguishing lights. In some experiments conducted by the authors, the oxygen content fell to 16.3 per cent before the flame became extinguished, but the presence of 10 per cent of carbon dioxide raised the extinguishing percentage of oxygen to 17.3.

Atmospheres that do not contain enough oxygen to support an oil-fed flame (about 17 per cent) may be explosive when the oxygen content is as low as 14 per cent, if enough methane is present.

When a burning part of a mine has been successfully sealed the composition of the atmosphere within changes. The oxygen decreases to a proportion (probably about 17 per cent) that will not support flame; ultimately the oxygen content becomes so small that the rate of combustion is extremely low, so low that combustion entirely ceases, the embers cool, and the admission of air when the mine is reopened does not rekindle them.

In 111 samples of gas from 29 mines represented, the average percentage of carbon dioxide in the black damp was 11.5 per cent, and the average percentage of nitrogen 89.5 per cent.

In 6 mines of 22 examined the temperature was higher than it should be (75° F., wet bulb) under the best ventilating conditions.

Analyses of a large number of samples, show how mine air changes as it traverses the workings. The average composition of the black damp was 9.2 per cent carbon dioxide and 90.8 per cent nitrogen. Except for two or three samples, in which carbon dioxide was high and the oxygen low, the quality of the air was good.

As regards the unfavorable effect of black damp on men, on lights, and on the explosibility of methane-air mixtures, the diminution of oxygen in the atmosphere, resulting in the formation of more nitrogen, is mainly responsible. The presence of carbon dioxide is far less

important; hence the objection to making the terms "black damp" and "carbon dioxide" synonymous.

Many of the mines of the Cripple Creek region are menaced with gas that is loosely held in the rock strata and that issues at times into the mines, so that workmen can not enter certain drifts and occasionally a whole mine.

This gas, according to Lingren and Ransome, is of deep-seated origin and probably represents the last exhalations of the extinct Cripple Creek volcano. It contains about 14 per cent carbon dioxide and 86 per cent nitrogen. The gas is confined in the rock strata under very low pressure, so that changes in outside atmospheric pressures affect its outflow.

A limited number of observations made by the authors indicate that the direction of the wind influences the outflow of the gas.

Data regarding ventilation conditions in 15 metal mines other than the Cripple Creek mines are shown. The average percentage of black damp found in these mines was 2.67 per cent. The average composition of the black damp was 9.3 per cent carbon dioxide and 90.7 per cent nitrogen, or about the same as that found in coal mines. In general, the composition of the air in these metal mines, as shown by the carbon dioxide and oxygen content, were good. Difficulty is experienced, however, in keeping down the wet and the dry bulb temperature. In many cases the temperatures were so high as to be detrimental to health.

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PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF COAL MINING.

Limited editions of the following Bureau of Mines publications are temporarily available for free distribution. Requests for all publications can not be granted, and applicants should select only those publications that are of especial interest to them. All requests for publications should be addressed to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

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DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

THE TECHNOLOGY OF MARBLE QUARRYING

BY

OLIVER BOWLES

WASHINGTON
GOVERNMENT PRINTING OFFICE
1916

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First edition. March, 1916.

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ILLUSTRATIONS.

PREFACE.

In 1914 an agreement was entered into, between the United States Geological Survey, the United States Bureau of Standards, and the Bureau of Mines, for a cooperative study of the building-stone industry of the country. In general, this agreement provided that the Geological Survey should study the data, compare the classification, extent, and geology of undeveloped and quarried deposits of stone and collect statistical information on production and value of output; that the Bureau of Standards should undertake all the physical and chemical tests required for determining the value of stone for structural purposes, or as aggregate for concrete, and that the Bureau of Mines should investigate all mining and technologic data, with special reference to safety, efficiency of operation, and prevention of waste. The general object of the cooperative agreement was to obtain comprehensive data on the occurrence, quality, and methods of preparation for the market of the various building stones of the United States. By agreement the work was begun with an investigation of the marble-quarrying industry.

The Bureau of Mines was fortunate in procuring for its share of the work the services of Mr. Oliver Bowles who had had experience in examining and describing quarries in Minnesota.

During 1914 Mr. Bowles personally visited 64 active marble quarries. Through the cooperation of quarry owners numerous representative samples of the marbles of the United States were obtained and shipped to the Bureau of Standards for testing. The results of these tests will appear in due time. The description of the investigations of quarry methods and the general technology of marble are published herewith.

The aim of this bulletin is to present to the marble workers of the United States a concise statement of the most efficient and economical methods now in use for producing and preparing marble. Progressive quarrymen realize the value of observing the methods employed by others in similar work, and it is hoped that the summary here presented will prove of use in their operations and will save them the inconvenience and expense of extensive travel to obtain such information for themselves.

The observations outlined refer chiefly to the methods employed in the quarrying of marble and the conditions that affect successful

operation; the structure of marble and its relation to quarry processes; waste through rock imperfections and inefficient quarry methods; and means of eliminating such waste or utilizing it if elimination is impossible. No attempt has been made to describe in detail all apparatus and equipment used in quarry operations, but special apparatus and any improvements or special designs tending toward increased efficiency are dealt with particularly.

Mr. Bowles has pointed out especially the great need of better systems of cost keeping as a means of keeping check on all unnecessary expense and as a means of testing the efficiency of methods and machines used in quarry operations. The importance of this feature can not be unduly magnified.

There is also pointed out a method for relieving earth pressure in certain quarries particularly subject to natural strains. This method promises to eliminate many of the fractures that constitute a prolific source of waste in quarry operations.

Operators of structural limestone quarries will find that many of their problems are discussed in this bulletin, as the methods and machinery they employ are similar to those used in quarrying marble. A more extended investigation of the particular problems of structural limestone and all other phases of limestone quarrying is contemplated.

Acknowledgment is made of many helpful suggestions received during the preparation of this bulletin. Too much emphasis can not be placed upon the valuable assistance rendered by Maj. J. S. Sewell, general manager of the Alabama Marble Co. His technical training, combined with practical experience, fitted him peculiarly for supplying just such information as the bureau desired and this information he imparted freely.

Others to whom acknowledgment is due are George R. Taylor, Marble, Colo.; J. P. McCluskey, Gantts Quarry, Ala.; A. W. Edson, Proctor, Vt.; John Kern and B. L. Pease, Knoxville, Tenn.; Prof. C. H. Gordon, University of Tennessee; Prof. F. J. Alway, University Farm, St. Paul, Minn.

The bureau desires to express its appreciation of the cordial treatment extended by quarrymen to its representative and the spirit of cooperation almost universally in evidence.

Charles L. Parsons,
Chief, Division of Mineral Technology.

THE TECHNOLOGY OF MARBLE QUARRYING.

By Oliver Bowles.

MARBLE IN GENERAL.

DEFINITION OF MARBLE.

In its geologic sense the term marble is applied to rocks consisting of crystallized grains of calcite or dolomite or a mixture of the two. Although limestone has the same chemical composition as marble it differs in that the component particles of calcium or magnesium carbonates are granular and noncrystalline. In marble the crystals may be intimately intergrown, whereas limestone is an aggregation of unrelated particles cemented together into a solid mass.

In its commercial sense the term marble has a much wider application. As susceptibility to polish is one of its chief commercial assets, all calcareous rocks capable of polish are classed as marbles. Limestones that show little crystalline structure may, if they take a good polish, be classed as marbles. Furthermore, serpentine rocks, even if they contain little calcium or magnesium carbonate, are classed as marbles, as they are commercial substitutes of true marbles.

COMPOSITION OF MARBLE.

Aside from serpentine and other extraordinary varieties, marble is made up almost entirely of calcium or magnesium carbonates. A calcite marble may consist of 96 to 99 per cent calcium carbonate. A dolomite marble, if impurities are disregarded, contains approximately 54 per cent calcium carbonate and 46 per cent magnesium carbonate. Marbles consisting of mixtures of calcite and dolomite may have compositions anywhere between these two extremes. The extremes may be illustrated by examples mentioned by Dale.[a] Marble quarried near Proctor, Vt., contained 98.37 per cent calcium carbonate, and a dolomite marble from Lee, Mass., contained 54.05 per cent calcium carbonate and 45.93 per cent magnesium carbonate. An intermediate type is represented by the crystalline magnesium limestone of Tuckahoe, N. Y., which contained 70.1 per cent calcium carbonate and 25.40 per cent magnesium carbonate.[b]

[a] Dale, T. N., The commercial marbles of western Vermont; U. S. Geol. Survey Bull. 521, 1912, p. 13.
[b] Kemp, J. F., A handbook of rocks, 1906, p. 138.

A varying percentage of chemical impurities is present in practically all marbles. The more common of these are silica (SiO_2), iron oxides (FeO and F_2O_3), manganese oxide (MnO), alumina (Al_2O_3), and sulphur; less common are minute quantities of the oxides of titanium, potash, sodium, lithium, and phosphorus. Organic matter is commonly present.

The impurities of marble are present in the form of grains of definite minerals. In some specimens the individual grains may be too minute to be recognizable with the naked eye, and in others they may attain considerable size. The more common mineral impurities are quartz (or some other form of silica, such as chert or flint), hematite, limonite, graphite, mica, chlorite, tremolite, wollastonite, diopside, hornblende, tourmaline, pyrite, or marcasite. In the marbles of southern Ontario, Parks [a] notes the occurrence of 37 minerals that have been formed by metamorphic processes acting on the impurities of the original limestone. The more common are quartz or some other form of silicon dioxide, pyrite, marcasite, mica, or chlorite.

Most marbles of commercial value contain small percentages of impurities.

ORIGIN OF MARBLE.

FORMATION OF THE ORIGINAL LIMESTONE.

Marble is derived from beds of limestone. The latter are formed in the sea, mainly as accumulations of calcareous remains of marine organisms, such as corals, rhizopods, and algæ. Water containing carbon dioxide is capable of dissolving calcium carbonate from the rocks through or over which it flows, and in consequence the water of rivers is charged with lime carbonate as it enters the ocean. Thus a supply of dissolved calcium carbonate is always at hand from which the organisms may manufacture their shells. Countless generations live and die and as a consequence the calcareous accumulation may be of vast extent. In places, the chemical precipitation of calcium carbonate may add to this accumulation.

There is abundant evidence that many limestones are of organic origin, as some of them are merely aggregates of fairly well preserved shells. In most specimens, however, a few fragmentary shells only remain in recognizable form, all others, through the beating of the waves or other activities, having been broken into minute fragments.

By pressure of superincumbent material and by deposition of some form of cement in the intergranular spaces the mass is later consolidated as a firm and coherent rock which is termed "limestone." Beds hundreds and even thousands of feet in thickness have been formed by such processes.

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[a] Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, p. 307.

METAMORPHISM OF LIMESTONE.

Marble is regarded as being the product of the metamorphism of limestone beds. That granular noncrystalline limestone can be changed into crystalline limestone or marble has been proven in the laboratory, as shown by Clarke.[a] From the results of various experiments he concludes that pressure alone, heat alone, or both together may result in the recrystallization. It is probable that the presence of water assists the process. Marble may therefore result from great pressure exerted on the strata by folding, or by heat produced from an igneous intrusion, or both agencies may work in conjunction. Recrystallization as a result of igneous intrusion has been observed by several authors.[b]

ORIGIN OF ONYX MARBLES.

Onyx marbles have a history rather distinct from that of the true marbles. Although consisting essentially of calcium carbonate, they are purely chemical deposits and have not resulted from the metamorphism of preexisting limestone beds. As pointed out by Merrill,[c] who gives a lengthy discussion of their origin and occurrence, they are of two types. One is a product of precipitation from hot springs, a travertine; the other is a deposit from cold-water solutions in limestone caves. Most deposits of onyx are formed in successive layers. Impurities such as iron and manganese oxides may be present in varying amounts in successive layers, and thus a beautiful banding may result. From the nature of their formation onyx deposits are necessarily limited in extent as compared with deposits of true marbles.

ORIGIN OF VERD ANTIQUE.

Verd antique or serpentine marble is in no respect comparable with true marble either in composition or in origin. Serpentine is in general derived from the alteration of basic igneous rocks such as the peridotites, which are rich in olivine and pyroxenes, or from magnesium silicate rocks formed by the metamorphism of limestone. The process is accompanied by hydration, with an addition of 13 to 14 per cent of water. The movement occasioned by the swelling that results probably accounts for most of the unsoundness common to verd antique.

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[a] Clarke, F. W., The data of geochemistry: U. S. Geol. Survey Bull. 491, 1911, pp. 531-532.

[b] Renwick, W. G., Marble and marble working, 1909, p. 4; Conybeare, W., Descriptive notes on the northeast coast of Ireland. Trans. Geol. Soc. London, vol. 3, 1816, p. 210.

[c] Merrill, G. P., Stones for building and decoration, 1903, pp. 242-296.

PHYSICAL PROPERTIES OF MARBLE.

HARDNESS.

Hardness may be defined as the resistance that the surface of a substance offers to abrasion. The hardness of calcite is given as 3 in Moh's scale, and that of dolomite as 3.5 to 4, whereas that of glass is about 5. The hardness of a marble as a whole may be different from that of the individual grains that compose it. The hardness is influenced by the degree of cohesion between the grains. Most fine-grained, compact marbles are harder than coarse-grained varieties. Some marbles are remarkably hard even if no silica or other excessively hard impurities are present. Hardness of the mass as a whole is an indication of "workability," and is an important property, as the cost of quarrying marbles that are worked slowly by tools is much higher than the cost of quarrying those easily worked. Although the cost of quarrying hard marble may be high, the hardness is a valuable property if the material is to be exposed to abrasion.

High resistance to abrasion is desirable in marbles that are to be used for sills, steps, or floor tile, all of which are exposed to the friction of the feet of pedestrians. Marble employed for such uses should be hard, and uniformity in hardness is desirable; otherwise the surface will soon become uneven. In constructing floor patterns of different marbles, it is important that the several varieties should be equally resistant to abrasion, as otherwise the floor will eventually become uneven. This condition may be observed in the floor of the Union Station at Washington, D. C., where the white tiles of Vermont marble, after eight years' use, are in places worn down nearly half an inch lower than the small squares of harder material from Swanton, Vt.

A second agent of abrasion is wind. Wind polish of a pronounced character has been observed on rocks much harder than marble. The Sioux quartzite of southwestern Minnesota, a rock that is probably harder than any other in the United States used for building purposes, has been so wind worn that corners have been rounded and the exposed surfaces have been given a glassy polish. Dust and sand carried by the wind on city streets tend slowly to wear away surfaces, mainly by removing insecure grains and thus exposing fresh surfaces to the agencies of weathering. The effects are most pronounced on corners and in narrow spaces between buildings where the force of the wind is concentrated. Egleston [a] states that in New York City many tombstones that face the prevailing winds are so worn that inscriptions are almost illegible.

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[a] Egleston, J., The cause and prevention of decay in building stones: Am. Architect, vol. 18, September 5, 1885, p. 113.

SPECIFIC GRAVITY AND WEIGHT PER CUBIC FOOT.

The specific gravity of a substance is its weight compared with the weight of an equal volume of water. The specific gravity of calcite is 2.7 and that of dolomite about 2.9. Consequently, dolomite marbles are heavier than calcite marbles. It is found that the actual weight per cubic foot of a block of marble differs more or less from its theoretical weight calculated from the specific gravity of the constituent minerals. A porous rock of given volume will be lighter than an equal volume of similar material that is nonporous. In most marbles the pore space is small, and the actual weight does not differ greatly from that calculated from the specific gravity.

The specific gravity of a compact homogeneous substance having no pore space may be determined in a simple manner, as follows: A thoroughly dried specimen is suspended from a balance by a thread and weighed in the air. It is then weighed while immersed in water. On account of the buoying up of the water it will weigh less while immersed. The loss in weight when immersed is the weight of a quantity of water equal in volume to the substance immersed. The specific gravity is, therefore, the weight of the substance in air divided by its loss in weight when immersed in water, as expresed by the formula $\frac{A}{A-B}$, in which A represents the weight of the substance in air, and B represents the weight of the substance when immersed in water.

An accurate determination of the specific gravity of a marble is, however, a less simple matter, on account of the pore space involved. In order to determine specific gravity accurately the pore space must be eliminated. This may be accomplished by the following procedure: Dry the rock specimen at 110° C. until all the moisture has been driven from the pores; then determine the dry weight. Next completely fill the pores with water, as by soaking the blocks in water for several weeks. Buckley,[a] accomplished the desired result by boiling the specimens for 36 hours under the receiver of an air pump that reduced the pressure to one-twelfth of an atmosphere. By such means the removal of air from all the pores may be facilitated. With the pores thus filled with water, weigh the specimen when immersed in water and determine the specific gravity as described above.

A more accurate method if care is exercised is to eliminate the pores by grinding the rock to a fine powder, and to determine the specific gravity of the powder. This may be done by means of a specific gravity bottle, with an accurately ground stopper that projects upward as a hollow tube of small diameter. The bottle is

[a] Buckley, E. R., Building and ornamental stones of Wisconsin: Wisconsin Geol. and Nat. Hist. Survey Bull. 4, 1898, pp. 65-67.

dried carefully and weighed. It is then filled with distilled water exactly to the top of the capillary tube and weighed again. Ordinarily the next step is to thoroughly dry the bottle, place within it a part of the finely powdered rock, and weigh. The difference between this weight and that of the empty bottle is the weight of rock taken. The rock powder is retained in the bottle, which is then filled with water and weighed again.

Certain mechanical difficulties make desirable a modification of the last two steps. Considerable time is required to thoroughly dry the narrow-necked bottle. After the dry mineral has been weighed and the bottle filled with water, it is difficult to prevent small particles from floating to the surface and flowing away with the superfluous water when the stopper is inserted. This small error may be avoided by reversing the last two steps. When the bottle full of water is weighed, the bulk of the water may be thrown out, the rock powder poured in, and enough water added to completely immerse it. After all bubbles have been removed the bottle may be filled with water to the top of the capillary tube as before and weighed. The loss of any fine particles during the process of filling and inserting the stopper does not affect the result, as the weighing is done subsequently. After the weighing has been completed, if the powder is thoroughly settled, and the water above it clear, most of the water may be carefully drawn off with a pipette. The remainder may be evaporated in a hot air bath, and the drying continued until the weight is constant. Thus the weight of the bottle containing the dry rock powder is the last figure obtained.

EXAMPLE.

If X = weight of dry bottle
Y = weight of bottle full of water
Z = weight of bottle containing powdered rock and filled with water
and A = weight of dry bottle and powdered rock
then A − X = the weight of the rock employed = W.

The sum of W and Y gives the total weight of rock, bottle, and water when none of the water is displaced. When the mineral is placed in the bottle and its volume of water is displaced the weight Z results. Therefore the weight of water displaced by the rock is $Y+W-Z=M$. The specific gravity is therefore $\frac{W}{M}$.

Marbles range in actual weight from 165 to 180 pounds per cubic foot. The economic significance of weight is chiefly in connection with the necessary strength of equipment for handling and freight charges for transportation. By knowing the average weight per cubic foot the quarryman may measure a block and calculate its weight with reasonable accuracy. He is thus enabled to judge the risk involved in handling it with any given equipment. Marble is a heavy structural material, and the necessary transportation charges

must be carefully considered when bidding on contracts at a distance from the point of production.

SOLUBILITY.

Calcium and magnesium carbonates are practically insoluble in pure water. Certain dissolved gases, notably carbonic acid gas, which are present in surface water in small proportions, render the water capable of dissolving the carbonates to a limited degree. Marbles exposed to the weather are therefore slowly dissolved. Although the process is slow, its effects may be considerable when long periods of time are involved. That marbles are more soluble than rocks consisting of silicate minerals is demonstrated in nature. Most marble deposits in humid regions are found in valleys formed by the more rapid erosion of the marble belts than of the bordering siliceous rocks.

The rate of solution is variable in different marbles. It depends on the chemical composition, texture, and porosity of the marble, the climate of the region, and the nature of the atmosphere. Near large cities various acids from smoke abound in the air and are taken up by rain water, thus increasing its power of solution. If a rock is permeable it dissolves more rapidly than if impervious. Calcite dissolves more rapidly than dolomite under the same given conditions if the texture of each is similar, but the tendency for dolomite to occur with granular texture usually reverses the order of their solubility. The solubility of marble deserves careful consideration if its use for exterior purposes is contemplated.

COLOR.

The color of a marble is one of its most important physical properties. It is governed by the nature of the constituents. Marbles consisting of pure calcite or dolomite are white because these minerals are white. A serpentine marble is green because the prevailing mineral, serpentine, is green. Variations from the white color of a pure marble are due to admixtures of foreign substances. Such impurities may be uniformly distributed and thus give a uniform coloration, or they may be present in bands or streaks, giving clouded or otherwise nonuniform colors. Examples of nonuniform color distribution are the "Pocahontas" marble of Alabama, the variegated marbles of Vermont, and the "crow foot" structure, irregular dark lines, characteristic of the Tennessee deposits.

The causes of some colors in marbles are easily determined. The black and grayish shades are to be attributed to carbon, which is usually present as fine scales of graphite though amorphous in the "crow foot" of Tennessee marble. Red, pink, or reddish-brown

shades are due to the presence of manganese oxides (MnO) and (Mn_2O_3) or to hematite (F_2O_3). Yellow-brown, yellow, or cream colors are caused by minute grains of the hydrous oxide of iron, limonite ($2Fe_2O_33H_2O$). Dale[a] attributes the green color of certain Vermont marbles to the presence of sericite (fibrous potash mica), and the purplish tint of one of the dolomites of the Lake Champlain region to a mixture of hematite and magnetite. Referring to some of the less common colors, Dale comments as follows:

The more uncommon colors of marble are purplish, as in the Pavonazzo and Seravezza breccias imported from Italy, bright yellow, as in the "Giallo Antico" from North Africa, and orange yellow, as in some marbles from Norway. Among the uncommon combinations of colors is that of rose-pink and deep green in the "Leifset Gloire" from Norway.

The same author states[b] that the dolomite of Hancock and Mount Tabor, Vt., owes its buff color to the presence of siderite. Vogt[c] attributes the sky-blue, red, and orange tints of some Norwegian marbles to organic compounds. The "golden vein" of the Colorado-Yule marble is thought to be caused by the permeation of solutions bearing manganese or iron oxides. The green bands in certain parts of the same quarry are, according to Merrill,[d] caused by the presence of chrome-mica. Parks[e] describes green marbles in Ontario in which the color is due to needles of light-green actinolite. The pink color of one marble he attributes to "scattered flakes of brownish glistening mica." He states that brown colors are in many instances due to the presence of mica. He refers also[f] to a marble having gray dots due in part to some crystals being clear and others milky, and in part to the presence of fine graphite. The clear crystals are dolomite, and the milky crystals calcite. Solution emphasizes the dotted effect.

For certain purposes, it is desirable to have a distinct contrast between chiseled and polished surfaces. Such a contrast is especially desirable in headstones on which inscriptions are cut. The contrast is usually more pronounced in the colored, and less marked in the white marbles. A chiseled surface is opaque and somewhat granular, and reflects rather than absorbs the light. Hence it tends to appear white or light colored even if the stone is dark. When a face is polished the reflecting surfaces are removed, permitting the light

[a] Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 20.
[b] Dale, T. N., The calcite marble and dolomite of eastern Vermont: U. S. Geol. Survey Bull. 589, 1915, p. 54.
[c] Vogt, J. H. L., Norsk marmor, Kristiania, 1897, p. 354.
[d] Merrill, G. P., A report on the Colorado-Yule marble properties, 1914, p. 10.
[e] Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, pp. 323-324.
[f] Parks, W. A., Op. cit., p. 329.

to enter the crystals and be absorbed, causing the polished surface to appear much darker than the chiseled surface.

TRANSLUCENCE.

Marbles differ greatly in their capacity for transmitting light. The more translucent varieties, if fine grained, are best adapted for novelties or other ornamental purposes. Some marbles are waxy in appearance, and this property seems to be related to translucence. Dale[a] gives the depths to which certain foreign marbles will admit light. They are as follows: Best Pentelicon, 0.59 inch; Parian, 1.37 inches; Carrara statuary, 1.18 to 1.57 inches. The reputation of some marbles depends greatly upon this quality. As far as is known no figures have been obtained for the depth of light penetration into American marbles. Certain beds of Alabama marble are notably translucent. The same quality has been observed in marbles from Massachusetts, Vermont, and Colorado. Certain modes of artificial treatment are known to increase the translucence of marble. Usually the effects of such treatment are far less permanent than the material itself, and consequently are not to be recommended.

TEXTURE.

The grains of calcite and dolomite that make up a marble mass are crystalline and have a definite rhombohedral cleavage. They are mostly twinned. Both the cleavage and the twinning of each grain are independent as relating to other grains. The texture is usually about the same in all directions, though in some marbles an elongation of grains in one direction has been noted. This characteristic is discussed more fully under the heading "Rift." The degree of interlocking of grains, and other features of cohesion have a definite relation to crushing strength, porosity, and workability. In certain dolomitic marbles the grains of dolomite may differ greatly in size and shape from those of calcite. The difference in grain diameter between the larger and smaller grains in some marbles is rather marked, and in others is small. Uniformity of grain is desirable.

The size of grain is commonly described as fine, medium, or coarse. Such terms are indefinite, and may have quite different meanings with different individuals, the interpretation being dependent upon the range of texture experienced by the observer. In order to place texture upon an absolute basis Dale[b] graded the marbles of Ver-

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[a] Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 17.
[b] Dale, T. N., Op. cit., p 54.

mont into six classes, based upon the average grain diameter. The classification is shown in the following table:

Classification of Vermont marbles by grades of texture.

Grade.	Grain diameter.			
	Maximum.	Average.	General averages.	
			Millimeters.	Inches.
	Millimeters.	*Millimeters.*		
1, extra fine	0.2	0.05 to 0.10	0.06	0.0023
2, very fine	.5	.07 to .16	.10	.0039
3, fine	.75	.10 to .25	.12	.0047
4, medium	1.0	.12 to .31	.15	.0059
5, coarse	1.5	.20 to .60	.24	.0094
6, extra coarse	2.54	.30 to 1.35	.50	.0196

The texture of marble is influenced by impurities such as graphite, sericite, tremolite, actinolite, and mica; and also by the folding or plication of the beds. The latter may cause elongation of grain or granulation of certain parts of the mass.

RIFT OR GRAIN.

The terms rift and grain are used synonymously for the direction of easiest splitting in marble. The rift is usually parallel with the bedding. It is probably due to elongation of grain caused by pressure. Dale[a], whose microscopic study of marbles has been extensive, states that "in some marbles one or two axes of the grains are much longer than the others, and the longer axes of different grains are parallel, giving the rock a certain schistosity which is usually parallel to the bedding."

The rift may be emphasized by the presence of fibrous or platey minerals such as scales of mica or graphite or needles of actinolite. These usually occupy positions with their long axes parallel to the direction of grain elongation, and thus increase the tendency to split in that direction.

Rift is a property of marble that the quarryman should take into account in planning operations. By taking advantage of this ease of splitting, drill holes for wedging may be spaced much farther apart than if no rift exists.

POROSITY.

Porosity is the volume of pore space expressed as a percentage of the total volume of the rock mass. The pore space of marbles is usually much less than that of limestones and sandstones, and more

[a] Dale, T. N., Op. cit., p. 18.

than that of granites, though exceptions toward both extremes are known. It varies from 0.0002 to 0.4 per cent.

A method of determining pore space may be quoted from Parks,[a] as follows:

The specific gravity is determined as already indicated and the test piece, full of water, is weighed. The difference between this weight and the dry weight of the sample gives the weight of the included water. If this latter figure be multiplied by the specific gravity of the stone, we obtain an expression which represents the weight of the stone which would be required to fill the pores. If this amount be now added to the weight of the dry sample, the result is the weight of that sample, provided that there were no pore spaces. This weight divided into the weight of the stone required to fill the pores, and multiplied by 100, gives the percentage of pore space in the stone.

The difficulty of filling all the pores by saturation has already been referred to in considering specific-gravity determinations. A method is here suggested by which this difficulty may be avoided.

Three factors are necessary in order to determine porosity. They are specific gravity, dry weight, and volume of the material employed. The specific gravity may be determined by the pycnometer (specific-gravity bottle) method as already described. The volume and the dry weight may be determined from a smooth-faced cube of about 1-inch edge. Drying the cube at 110° C. until the weight is constant gives the dry weight. The specimen may then be coated with a thin film of paraffin, wax, or other waterproof substance to make it impervious, and its volume may be determined from water displacement in a large specific-gravity bottle. The film must be made as thin as possible, as its presence increases the volume of the block. If the quantity of waterproof substance used in coating the block is determined, a correction may be made by subtracting its volume from the total volume obtained.

Another method is to saturate the block and then determine its volume by water displacement without any coating. If it is even approximately saturated, further absorption during the brief time occupied in determining its water displacement will be negligible.

The method of calculating porosity is as follows:

If X = dry weight of block
G = specific gravity of material composing block
and V = volume of block
Then $V \times G$ = weight of block if all the pores were filled with marble = Y.
$Y - X$ = weight of marble required to fill pores = W

Percentage of pore space $= \frac{W}{Y} \times 100$.

Porosity is commonly expressed as "ratio of absorption," which is the percentage by weight that the absorbed water bears to the dry

[a] Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, p. 61.

weight of the stone. The specimen is dried in a hot-air bath at a temperature of about 110° C. until the weight is constant. The stone is then immersed in water for a period of time varying from two days to several weeks. The process of absorption may be assisted by boiling or by placing the immersed block beneath the receiver of an air pump as already described. The difference in weight between the dry and the saturated specimens is the weight of absorbed water. The ratio of absorption of American marbles as determined by the Bureau of Standards varies from 0.0018 to 0.00007.

Pore spaces in marbles permit infiltration of water, which may dissolve the stone, or may cause disintegration by freezing. The evils associated with porosity are discussed more fully under the heading "Imperfections of Marble."

STRENGTH.

The strength of a stone is the measure of its capacity to resist stresses of various kinds. It depends partly on the rift of the rock and on the cleavage and hardness of the grains, and partly on the state of aggregation, including degree of cohesion, interlocking of grains, and nature of cementing material, if such is present. Although strength alone is not a sure criterion of durability, a knowledge of the capability of any stone to withstand stresses of various kinds is of great value if the material is to be used for purposes involving extraordinary strains.

Many tests have been made of the strength of building stones. It was early learned from these tests that most stones have many times the strength required for ordinary uses. As pointed out by Buckley,[a] ordinary building stones have 2 to 10 times the crushing strength required in any structure for which they may be used. As a consequence of a recognition of this fact, there was a reaction against making tests, which were regarded as superfluous. An increased demand for strength in structural stone and a wider knowledge of the significance of strain resistance has lately led to a renewed interest in strength tests. It is known that stones are less durable when exposed to intense strains, and it seems reasonable to conclude that the rate of disintegration increases with proportional rapidity as the strain to which the rock is subjected approaches more and more nearly to the ultimate load it is capable of bearing. Rock strength may therefore have a decided influence on the rate of disintegration, even when it is evident that the strength is far in excess of the requirements.

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[a] Buckley, E. R., Building and ornamental stones of Wisconsin: Wis. Geol. and Nat. Hist. Survey Bull. 4, 1898, p. 59.

The tests commonly made are for crushing strength, transverse strength, elasticity, and shearing. As tests can be made only with high-priced specially designed equipment, which is available to few people, a description of the methods of testing is omitted, and the discussion confined to a brief consideration of the significance of the various types of stress resistance of which rocks are capable.

CRUSHING STRENGTH.

As already stated, most rocks that, after ordinary superficial inspection, would be chosen for structural purposes have many times the crushing strength required for ordinary uses. For certain purposes, however, such as bridge piers, abutments, columns, and the base blocks of very high structures, crushing strength demands more than ordinary attention. The tendency, more noticeable every year, to increase the height and superincumbent weight of great city structures makes strength tests more and more useful.

Rock structures have a definite influence on strength. As a rule rocks will bear a greater compressive stress across the bedding plane than parallel with the bedding plane. Hence stones should not be laid with the bedding planes vertical.

TRANSVERSE STRENGTH.

Transverse strength may be measured by testing the capability of a bar of stone supported at its ends to bear weight exerted at its center. Such tests indicate the suitability of a marble for door or window caps, or as bridging material that must bear a heavy load. Breakage of such caps, hovever, must not always be attributed to a weakness in the material employed, as unequal settling or improper laying may be the chief causes.

ELASTICITY.

When subjected to crushing strain rocks are capable of being appreciably compressed before rupture takes place. A measure of this compressibility in terms of the load is what is known as the modulus of elasticity. Parks [a] defines it more explicitly as follows: "The decrease in length of a bar of material thus subjected to pressure divided into the original length of the bar, and multiplied by the load in pounds per square inch, gives what is known as Young's modulus or the modulus of elasticity or compressibility." Merrill [b] found that after relief from intense pressure below the point of rupture rocks failed to completely recover their original form. This he termed a permanent "set."

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[a] Parks, W. A., Report on the building and ornamental stones of Canada, vol. 1, 1912, p. 47.
[b] Merrill, G. P., Stones for building and decoration, 1913, p. 478.

Compressibility may be the cause of cracks in the lower courses of certain large structures. It is evident that building stones having a low modulus of elasticity may under heavy superincumbent load be appreciably compressed with a resultant settling of the structure, and if one part of the building is composed of blocks having a different modulus of elasticity from those of an adjoining part, the settling will not be uniform. Such settling can take place only under extremely heavy loads. A knowledge of the elasticity of marble is, as quoted by Buckley,[a] "valuable in determining the effect of combining masonry and metal or of joining new masonry to old; in calculating the effect of loading a masonry arch; in proportioning abutments and piers of railroad bridges subject to shock, etc."

SHEARING.

The tendency to shear—that is, the tendency of one part of a block to slide laterally with respect to another part—is strong in certain structures, such as massive arches and lintels. Certain blocks in large buildings are subjected to strains in different directions, and the tendency to shear may be pronounced. Thus, shearing tests of marble designed for such purposes are of value.

WEATHERING OF MARBLE.

The term "weathering" is applied to the disintegration that results from exposure of rock to the various natural agencies that are active at or near the surface of the earth. In previous paragraphs a discussion is given of solubility, porosity, permeability, chemical composition, hardness, texture, and state of aggregation of marble.

The rate of decomposition by weathering is somewhat dependent on these physical features. For example, a soluble rock weathers more rapidly than one that is relatively insoluble, and an open-grained porous rock decomposes more readily than one that is more solid and impervious. Owing to this direct dependence of the rate of weathering on the physical properties of marbles, tests of the various qualities of a given stone are of great value in estimating its probable rate of weathering.

Climatic and atmospheric conditions greatly influence the rate of weathering. Oxygen and carbon dioxide are the most effective atmospheric agents of decomposition. The rate of weathering depends also on temperature. Little or no weathering takes place while the temperature is below 0° C. As the temperature rises, solution takes place with increasing speed and in tropical regions is active throughout the entire year. Rapid changes in temperature

[a] Buckley, E. R., Building and ornamental stones of Wisconsin: Wis. Geol. and Nat. Hist. Survey Bull. 4, 1898, pp. 63-64.

cause rapid disintegration, especially in regions where chemical action is supplemented by the effects of frost.

Aside from the effects of frost, changes in temperature produce differential expansion and contraction, which set up shearing stresses, causing flakes to split off. Rupture by expansion and contraction is probably less in the porous than in the nonporous rocks, as in the former case necessary adjustment is made between the grains. As a rule the finer-grained rocks weather less rapidly than do those of coarser grain. Van Hise [a] states that "this is a consequence of the closer interlocking of the mineral particles of the fine-grained rocks, and of the fact that the differential expansion and contraction by change of temperature is less with fine particles than with coarse particles."

Humidity greatly favors decomposition, as chemical action is slow in the absence of water. For example, Cleopatra's Needle, which stood thousands of years in arid Egypt, began to disintegrate so rapidly in Central Park, New York City, that it had to be coated with paraffin to prevent destruction.

The most favorable conditions, therefore, for rapid weathering are coarse and permeable texture of the stone, humidity, warmth, and rapid changes of temperature between points above and below freezing.

Parks [b] conducted a series of interesting experiments designed to test the relative durability of various stones. As carbon dioxide is the most active agent of chemical decomposition, its effect on various stones was taken as an indication of their relative durability. Cubes of stone were immersed in distilled water through which a stream of carbon dioxide was passed, and the corrosive effect was tested by determining the loss in weight by solution in a given time. Those cubes that showed a comparatively small loss by solution were regarded as the most durable.

The change in color of the rock surfaces during the process corresponded closely with the color modification brought about by many years of weathering. Hence such experiments are useful as a means of testing the probable color changes in untried rocks intended for exterior structural work.

Dale [c] investigated the rate of disintegration of marble, especially that used for tombstones. He states that in New England the lettering on tombstones 75 to 100 years old will probably be entirely effaced in 300 years from the date of cutting. He noted a marble slab at Plymouth, Mass., on which the lettering was almost effaced after 87 years of exposure. He mentions also a block of South

[a] Van Hise, C. R., A treatise on metamorphism: U. S. Geol. Surv. Mon. 47, 1904, p. 533.

[b] Parks, W. A., Op. cit., pp. 70-71.

[c] Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 38.

Dorset (Vt.) marble on which the edges of the letters are fairly sharp after 79 years' exposure.

Although tests of the various physical properties of marble are useful as an index of its probable durability, actual observations of old structures, outcrops, or quarry walls give the most dependable information. Observations may be made of the rate and uniformity of weathering and the stains or other changes in color involved.

Weathered outcrops give the marble prospector valuable information. On ledges where successive beds are exposed the most enduring beds will stand out in relief. The condition of soundness can also be easily recognized, as weathering emphasizes all cracks or lines of weakness.

Erosion of marble beds commonly results in the formation of "mud pockets," a name given to cavities worn in the rock and later filled with soil. In some places they are large, attaining depths of 50 or 60 feet, and the removal of the clay contained in them is a matter of considerable expense to the quarry operator. In the Tennessee marble deposits erosion has followed the unsoundness and has left numerous masses of rock projecting upward from the solid beds and surrounded by clay. These constitute the so-called "bowlder quarries."

Most serpentine marbles are veined with calcite or dolomite, and hence weather unevenly. Moreover, most of them are unsound, and consequently permeable to rain water. Such marbles lose their polish, weather rapidly, become stained, and soon lose both attractiveness and strength if exposed to atmospheric agencies.

VARIETIES OF MARBLE.

Marbles may be classed as statuary, decorative, building, and monumental.

Statuary marble is the most valuable variety quarried. It must be pure white in color, uniform and fine-grained in texture, and somewhat translucent, and must possess a marked adaptability for carving. Renwick[a] recommends that "marble for statuary purposes should never be selected in bright weather. Veinings and discolorations are more difficult to discover at this time than at any other. A dull day with a good light is the best time for inspections; if after a shower of rain, so much the better. Provided no rain has fallen, the blocks should be soused with water. Veins and stains can then be more readily perceived. If possible have each block slung and struck with a hammer. If the sound of the blow is dull and heavy, look out for cracks. Should a hard and metallic tone be emitted, the marble will be heavy in working; but if a soft, clear ring is heard, the material is sound and will both work and wear well."

[a] Renwick, W. G., Marble and marble working, London, 1909, p. 61.

Decorative marble is usually employed in places unexposed to the weather, and hence may be selected for appearance without regard for the effects of weathering. Marble containing large amounts of iron sulphides may therefore be used for interior decoration, and such marbles may give beautiful effects. Brecciated marbles consisting of angular fragments cemented together in nature by calcium carbonate are widely used for columns. Most marbles of this type are imported, only a limited amount having been produced in the United States up to the present. Both pure white and variously colored marbles that possess unusual attractiveness may be used for interior decoration.

Verd antique, or serpentine marble, is in common use. Most of such marble is somewhat unsound and not of sufficient strength to justify its use where it will have to take a heavy load. The marble is commonly used for exterior decoration, but is not to be recommended for this purpose, as it does not weather uniformly and soon loses it polish. For interior decorative purposes it is popular.

Onyx marble possesses a waxlike luster and an attractive banding which make it a popular material for interior decoration.

Numerous statuary and decorative marbles from American quarries are now on the market. No two are alike, and each has its own particular trade name.

Marbles for building and monumental purposes must have attractive and uniform colors, and in addition must possess the ability to withstand weathering and to retain their attractive appearance. For interior decorative marble appearance is the prime factor determining its value, whereas with exterior marble qualities of endurance rank equally in importance with appearance. Building marble should therefore be strong, uniform, close-grained, reasonably nonabsorptive, and free from such impurities as may stain or corrode the surface.

DISTRIBUTION AND PRODUCTION OF MARBLE IN THE UNITED STATES.

As stated in the discussion of the origin of marble, the recrystallization of the original limestone is promoted chiefly by heat and pressure. As a consequence most marble beds are confined to areas of extreme folding or igneous intrusions, and hence are to be found chiefly in mountainous regions. In some beds recrystallization takes place without igneous intrusion or movement of the beds, as at Kasota, Minn., but such conditions are rare. The important marble belts of the United States are found in the Appalachian region of the Eastern States and in the Rocky Mountain and the Coast Ranges of the West.

According to figures compiled by Burchard,[a] the States producing marble in 1913, arranged according to value of output, are as follows: Vermont, Tennessee, Georgia, Colorado, Alabama, Massachusetts, New York, Pennsylvania, Alaska, California, Maryland, North Carolina, Utah, Arkansas, New Mexico, Washington, Virginia, and Oregon. Deposits of onyx marbles occur in Arizona, Colorado, Utah, New Mexico, California, Kentucky, and Virginia. Deposits of serpentine marble, or verd antique, occur in Vermont, Pennsylvania, Georgia, Maryland, California, Massachusetts, Connecticut, Delaware, Maine, New York, New Jersey, New Mexico, and northern Michigan.

The value of marble produced in the United States in 1913 was $7,870,890, and of this amount the three States, Vermont, Tennessee, and Georgia, produced over 76 per cent. The average price per cubic foot was $2.11.

THE IMPERFECTIONS OF MARBLE.

UNSOUNDNESS.

MEANING OF UNSOUNDNESS.

When intersected by joints or fissures marble blocks are said to be unsound. The term "unsoundness" refers to all cracks or lines of weakness, other than bedding planes, that cause the marble to break before or during the process of manufacture. The various types of unsoundness are known locally as "joints," "headers," "cutters," "hair lines," "slicks," "seams," "slick seams," "dry seams," or "dries," and "cracks." The term "reed" is applied to a weakness parallel with the bedding.

IMPORTANCE OF JOINTS IN MARBLE DEPOSITS.

The presence of joints in marble deposits presents a most serious problem. They may be so close and irregular that the quarrying of profitable material is impossible. Joints should have a marked influence on the mode of quarrying a marble, in order that the waste due to their presence may be reduced to a minimum. The manner in which joints occur and their probable continuance at depth are matters of supreme importance.

NATURE OF JOINTS.

Most joints as they appear in marble deposits are straight and uniform though some may be curved or irregular. Some are open and conspicuous and others so obscure that they can be recognized only by those skilled in their detection by long and constant practice.

[a] Burchard, E. F., Stone: Mineral resources U. S. for 1913, U. S. Geol. Survey, 1914, p. 1313.

Becker[a] has pointed out that surface tension of water in joints tends to keep them closed. With a space of 0.01 inch the surface tension exerts a force of 13.5 pounds per square foot tending to draw the surfaces together, and if the opening is only 0.001 inch wide the force will be 135 pounds per square foot.

The most striking characteristic of joints is their tendency to occur in parallel systems. The occurrence of two systems approximately at right angles to each other is common. Occasionally a third or fourth system may appear. In exceptional cases joints may present such extreme irregularity that no well-defined system can be recognized.

The spacing of joints is variable. They have a tendency to occur in groups of closely spaced fractures separated by masses in which joints are few in number. In certain Vermont quarries such closely spaced groups are termed "fish-backs." In some deposits joints may be 10 to 30 feet apart, and in others they may be separated by a few inches only. Needless to say, a wide spacing adds greatly to the commercial value of a deposit.

ORIGIN OF JOINTS.

In order that one may even approximately understand the distribution, direction, spacing, and persistence at depth of the joints that intersect marble deposits a knowledge of their origin is necessary. Authors generally agree that joints are caused by strains in the rock masses. It is thought that few joints are due to tensile stresses, as joints so caused would show no slickensided surfaces, and be irregular in form, whereas most joints are straight and even planes and are somewhat slickensided. It is now generally accepted that practically all joints are faulted surfaces although the displacement may be small.[b] Daubrée was so firmly convinced of the correctness of this theory that he rejected the term "joint" as failing to imply the existence of relative motion, and introduced the terms "diaclase" and "paraclase." Joints are probably caused by pressure, and pressures in rocks may be highly complex. Curved joints indicate that the direction of effective pressure varies, or varied, from point to point.

The famous experiments of Daubrée[c] indicate that joints may be produced by simple pressure, the joint planes forming at angles of 45° with the line of pressure. His experiments also show that torsion may cause joints. Torsional strain in glass produced two sets of fractures approximately at right angles to one another and usually at angles of about 45° to the axis of torsion. Becker[d] gives

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[a] Becker, G. F., The torsional theory of joints: Trans. Am. Inst. Min. Eng., vol. 24, 1894, p. 131.
[b] See Becker, G. F., Loc. cit.
[c] Daubrée, A., Étude synthétiques de géologie expérimentale, Paris, 1878, pp. 300-374.
[d] Becker, G. F., Op. cit., p. 136.

a reason for this arrangement. He shows that torsion of a bar causes diagonal lines to be elongated or contracted, the directions of maximum extension and compression being 45° to the axis of torsion and perpendicular to each other. Cracks in fan-shaped groups may also result from torsion.

Crosby [a] claimed that earthquakes were important factors in the production of joints, each system being parallel with the earth waves producing it. Various systems must therefore have resulted from successive shocks. A second system will usually be nearly at right angles to the first, as an oblique shock would have found relief along the fractures of the first system. The presence of an oblique system presumes a shock of such high velocity that time was not allowed for the strain to find relief along previous fractures. The same author in a later publication [b] shows convincingly how shock and torsion may act in conjunction and produce results according with the phenomena as they occur in nature. His theory, in brief, is as follows:

The torsional theory assumes a very slow, and the earthquake theory a rapid process of joint development. As fractures formed by slow processes are apt to be irregular, following all places of weakness, torsional joints should be irregular. Most joints, however, are fairly regular and are even known to pass directly through the hard pebbles of a conglomerate. Thus, a discrepancy appears in the torsional theory. A single system of regular joints may therefore be ascribed to shock. A subsequent shock in a transverse direction would tend to break up the sheets formed by the joints, the fractures being of a less continuous nature than those produced by the first shock, being terminated abruptly in many places by the first system of joints and continuing in a different though parallel plane. Thus two systems of an unlike character are best explained by the earthquake theory. However, two systems of like character may be referred to the torsional theory except when they exhibit the regularity indicative of instantaneous stresses.

Crosby refers to the idea, upheld by many geologists, of an almost universal condition of strain in the earth's crust, and states that if, while under torsional or folding stresses, the rock is traversed by an earthquake wave, fractures may be precipitated. Experiments with sheets of glass showed that a sudden shock while the torsional strain was distinctly below the breaking point would precipitate the fractures, but that the direction of the fractures was governed by the direction of the axis of torsion. It seems reasonable, therefore, that the result of an earthquake wave traversing rocks under strain would

[a] Crosby, W. O., On the classification and origin of joint structures: Proc. Boston Soc. Nat. Hist., vol. 22, 1882–83, pp. 72–85.

[b] Crosby, W. O., The origin of parallel and intersecting joints: Am. Geol., vol. 12, 1893, pp. 368–375.

be the sudden development of joints, governed in direction by the torsional or folding stresses present.

Becker[a] presents some instructive figures showing the manner in which joints are developed by a compressive force in one direction. In figure 1, A, the force is supposed to act in a direction at right angles to the upper and lower surfaces. Face *x* and its opposite face are supported to prevent rupture. The force produces fractures in planes perpendicular to *x* and inclined in opposite directions at angles of 45° from the line of force. The face *x* is intersected by two systems of joints at right angles to each other. On the other four faces perpendicular to *x* they appear as parallel lines but may dip in either of two directions. If considerable deformation takes place before rup-

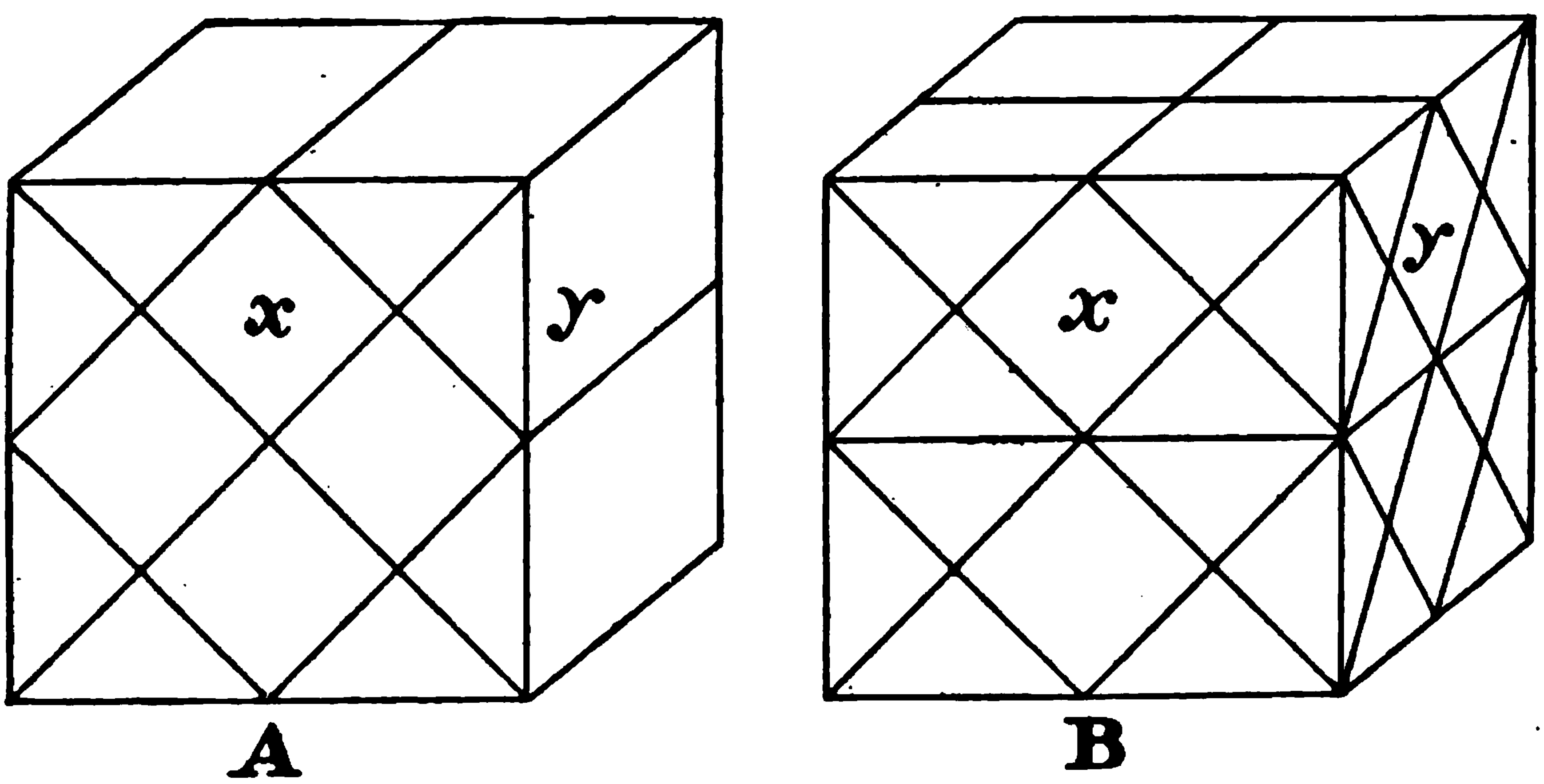

FIGURE 1.—Diagrams illustrating the manner in which a single force may produce several systems of joints—A, joints of two systems, B, joints of four systems.

ture, the joints will make angles of more than 45° with the line of force.

If the support on *x* or *y* is the same, or, in other words, if the resistance perpendicular to the line of force is uniform, two systems will form simultaneously, as shown in figure 1, B, and horizontal and diagonal cracks will appear on both *x* and *y*. Thus four systems of joints parallel with octahedral faces will be produced, and the resulting blocks will be octahedral or tetrahedral in form. Other figures given by the author indicate how such systems may be shown to appear on a random plane.

It has been shown in the preceding paragraphs that torsional forces, compressive forces with uniform or nonuniform relief in a transverse direction, or earthquake shocks alone or in conjunction with other forces may produce definite systems of joints. The forces may be

[a] Becker, G. F., Simultaneous joints: Proc. Wash. Acad. Sci., vol. 7, 1905, pp. 267-275.

multiplied and complex, and the resulting joint systems may present a corresponding complexity, but definite systems are generally developed.

The nature of joint surfaces and some inferences therefrom have been discussed by Woodworth.[a]

PRACTICAL ILLUSTRATIONS OF JOINT SYSTEMS.

The occurrence of joints as observed in many quarries clearly supports the conclusion deduced by theory in so far as systematic arrangement is concerned. A few examples selected from many may be offered in support of this conclusion.

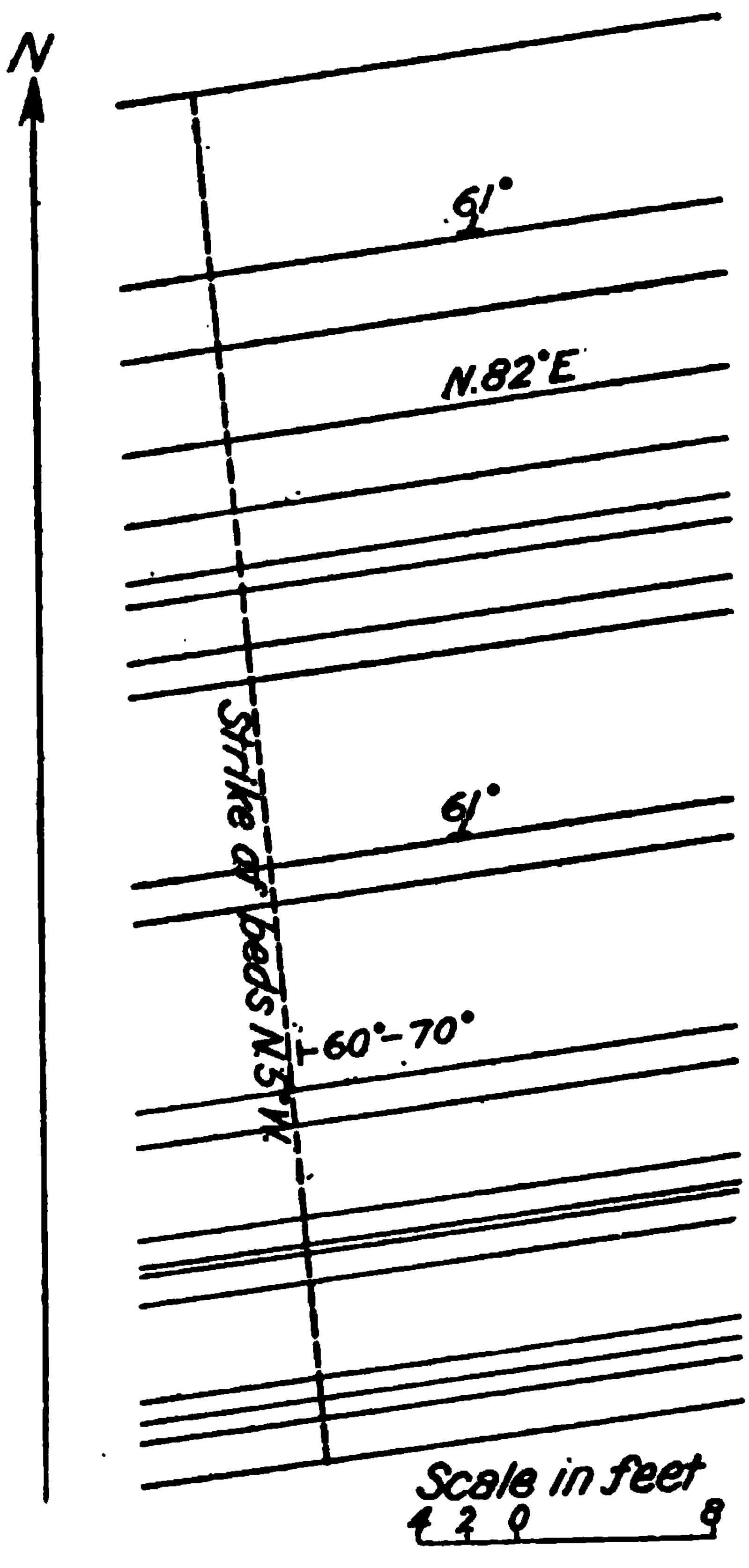

FIGURE 2.—A remarkable system of 21 parallel joints, as found in one outcrop.

In many quarries jointing systems are clear and definite. The direction and spacing of joints as observed at the surface may persist with remarkable uniformity at depths of 100 feet or more. Figure 2 illustrates a system of 21 parallel joints drawn to scale as they appear in an outcrop at a quarry in West Rutland, Vt. They strike N. 82° E. and dip 61° N. 8° W. At a depth of 145 feet they have the same strike, dip, and spacing as at the surface. Four of them are shown in Plate I, *A*.

Figure 3 shows open joints as they appear in a quarry wall. The horizontal lines represent the various floors cut out by channeling machines. The bedding dips about 40° and is marked by one prominent and several less conspicuous open seams. Although some irregularities are to be noted, a definite jointing system may be recognized.

Plate I, *B*, illustrates a system of parallel joints in a Tennessee marble deposit. Erosion has followed the joint planes and thereby given them exceptional prominence. Attention is directed to figure 11 (p. 67) which shows the joint systems in the floor of a marble quarry in Alabama.

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[a] Woodworth, J. B., On the fracture system of joints, with remarks on certain great fractures. Proc. Bos. Soc. Nat. Hist., vol. 27, 1896, pp. 165-183.

A. THE ABRUPT TERMINATION OF JOINTS IN A PLASTIC LAYER IN WHICH DEFORMATION BY FLOWAGE TAKES THE PLACE OF FRACTURES.

B. A SYSTEM OF PARALLEL JOINTS EMPHASIZED BY EROSION.

Hundreds of examples might be given of joint systems, and many quarrymen could supplement them with illustrations from their own experience. On the other hand, in some quarries joint systems may be difficult to recognize. They may be obscure and seemingly rather irregular. However, careful mapping of many of them will reveal definite systems. A practical illustration may be of value.

The operators of a certain quarry declared that the joints in their quarry occurred without any definite system. In order to test the validity of this view the positions of the visible joints were determined as completely as circumstances would permit, with compass, clinometer, and tape measure. A plan, as represented in figure 4, was made showing the arrangement of the joints on the quarry floor. Although some irregularities may be noted, it is evident that only a limited number of joints vary more than a few degrees from a direc-

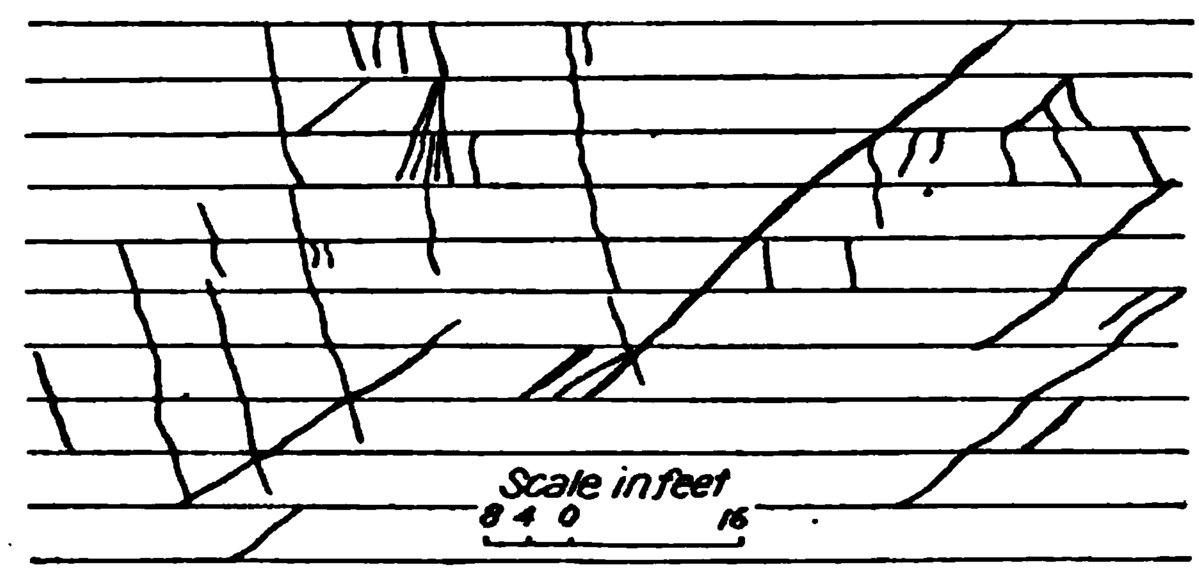

FIGURE 3.—Open joints in a quarry wall.

tion N. 45° W. While variations in dip are rather pronounced, a majority of the fractures are within limits of 10° from the vertical.

It is evident, therefore, that with few exceptions definite systems of joints may be recognized in marble quarries. The economic importance of joints is discussed in detail in a subsequent section devoted to a consideration of channeling in relation to unsoundness.

PERSISTENCE OF JOINTS AT DEPTH.

The disappearance or continuance of joints with depth is a matter of profound importance to quarrymen. The belief is common that joints are less numerous at depth. Some persons who hold this opinion can support it by observations in quarries, whereas with others the idea is the expression of an unverified optimism. The origin of joints and the phenomena accompanying their development have a direct bearing on their persistence at depth.

Becker[a] has shown that a mass of rock must occupy a greater space after jointing than before, as cracks and open corners are pro-

[a] Becker, G. F., Op. cit., pp. 274, 275.

duced. Thus as joints demand increased volume in the mass of rock affected, and as surface rocks have freedom of upward motion, whereas deep-seated rocks are more or less restrained in all directions, there is a stronger tendency for joints to form near the surface than at depth. As pointed out by the same author, the pressure on rocks at depth does not obviate the tendency for fractures to form, but may prevent actual ruptures. Deformation without rupture results, as is clearly shown in Plate I, *A*. The sudden termination of the joints in a plastic layer of soft green marble, as shown near the top of the plate, illustrates the manner in which plasticity permits the necessary adjustment by flowage rather than by fracture.

It is evident, however, that a condition of flowage demands an immense superincumbent load, and takes place, therefore, at depths far

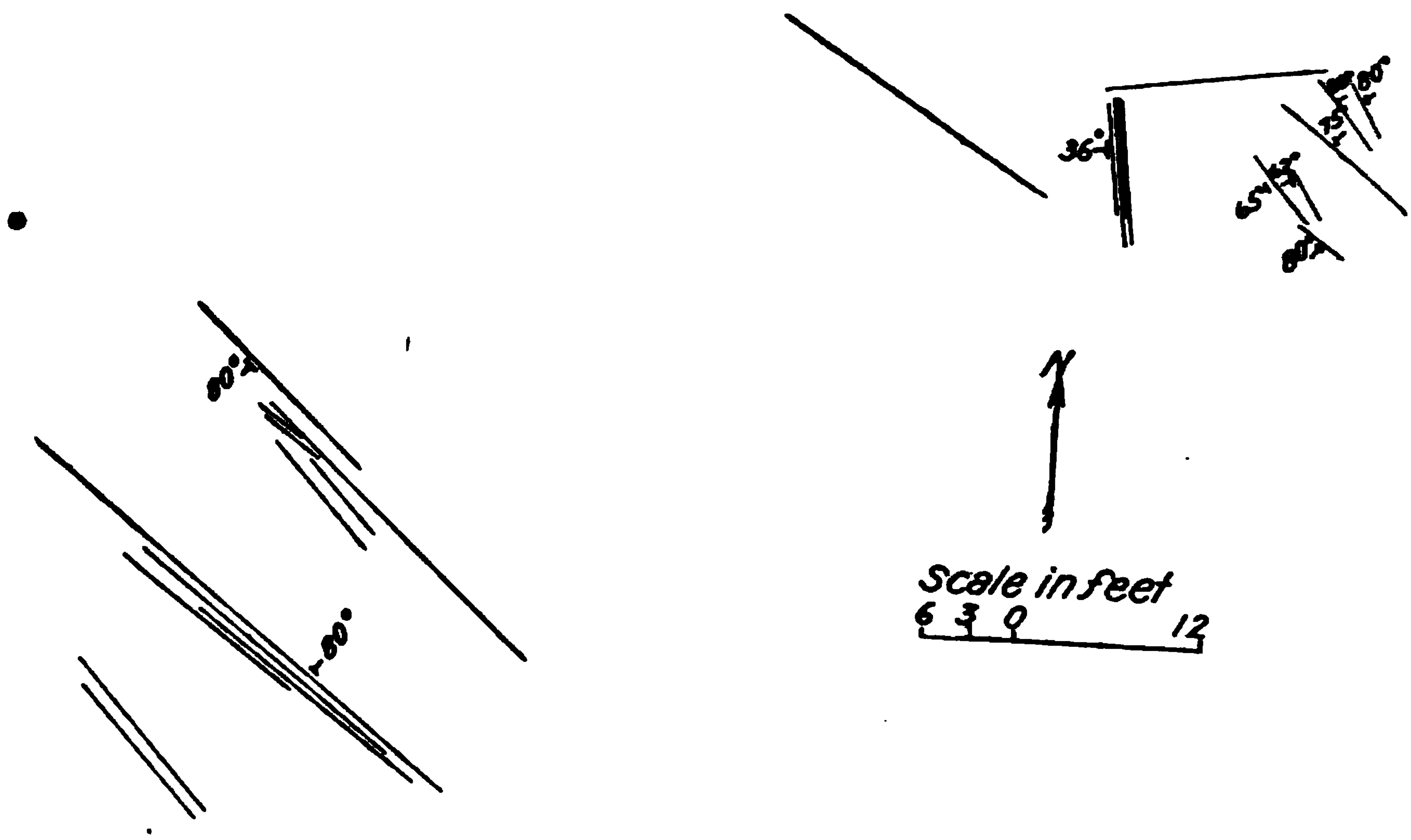

FIGURE 4.—Surface plan of joints that appeared to lack systematic arrangement.

beyond the reach of quarry excavations. If deformation without rupture takes place within the limited depth of a quarry pit, it must be by some other phenomenon than that commonly called flowage.

Deformation may take place at moderate depth by the formation of crushed structures which may be masked by recrystallization. Furthermore, deformation without rupture may take place by twinning, or the development of gliding planes in the calcite crystals. As gliding planes have been produced in the laboratory, it is evident that they may occur in nature under pressures of no greater magnitude than those brought about by a superincumbent load of rock within the vertical range of a quarry pit. Deformation may take place by one of these means at moderate depths.

In theory, therefore, joints are characteristically surface phenomena, and should decrease in number with increasing depth. This theory

is generally substantiated by the conditions disclosed in quarrying. The decrease may not be constant or uniform. Certain beds are more liable to be intersected with joints than other beds, and in consequence one bed may have more joints than those above it. In general it has been found, however, both in quarrying and in projecting core holes, that if any one steeply inclined bed is followed downward the unsoundness constantly decreases.

Certain fractures, known locally as "slicks" or "hair lines," are, as a rule, evenly spaced, vertical, and at right angles to the strike of steeply inclined beds. They usually disappear at depths of 50 to 100 feet. They are regarded as originating from the expansion and contraction of rocks owing to variations in solar heat.

UNSOUNDNESS IN VERD ANTIQUE.

Unsoundness in quarries of serpentine marble, or what is commonly called "verd antique," is usually rather pronounced and extremely irregular. It is probably caused chiefly by expansion or swelling, owing to the process of hydration as the serpentine is formed. As a consequence, no definite systems of cracks are to be expected. The formation of lens-shaped masses is common. It is frequently difficult to obtain blocks of any considerable size sufficiently coherent to be of commercial value. Occasionally the cracks are recemented by crystalline calcite, producing an attractive white veining on a green background. The so-called brecciated marbles are extremely unsound masses composed of many irregular and usually angular fragments that have been cemented by chemical precipitation of calcium carbonate.

"GLASS SEAMS."

In certain quarries joints that have been recemented in the manner described above are termed "glass seams." They may be sufficiently strong to permit sawing even into thin stock, but most of such seams are planes of weakness. The filling is most commonly of calcite. Occasionally it is of silica, either as quartz, flint, or chert. The silicious filling is least to be desired, as its extreme hardness makes sawing and polishing difficult, and it presents a nonuniform surface. In any case a glass seam usually appears as a conspicuous line which can be regarded only as a blemish when present in otherwise uniform marble.

IRON SULPHIDES.

THE COMMON IRON SULPHIDES.

The sulphides of iron which occur commonly in nature are pyrrhotite, pyrite, and marcasite. Pyrrhotite is a mineral of variable composition ranging from Fe_5S_6 to $Fe_{16}S_{17}$. It appears to be a solid

solution of FeS and sulphur. Pyrite and marcasite are identical in chemical composition, which is expressed by the formula FeS_2, but crystallize differently, pyrite being isometric, usually as cubes, octahedrons, or pyritehedrons, and marcasite orthorhombic, with prominent domes and pinacoids. In other respects the two minerals are closely similar and it is difficult to distinguish them.

Pyrrhotite is rare in marble. Pyrite and marcasite are accessory minerals in many marble deposits, the pyrite being the more common. They may appear as scattered crystals of variable size, or may form bands and masses of considerable prominence. The conditions under which pyrite, marcasite, or mixtures of the two may form in nature are discussed by Allen, Crenshaw, and Johnston.[a]

MANNER AND EFFECT OF SULPHIDE DECOMPOSITION.

By weathering, pyrite may combine with oxygen to form iron oxide, iron sulphate, or free sulphuric acid. The oxide may stain, and the acid may corrode the rock. When alumina is present whitish crusts of aluminum sulphate and alum may form. As pointed out by Merrill,[b] if magnesia is present the sulphuric acid may produce a soluble efflorescent salt, which forms white patches on the surface.

The process of weathering may, on the other hand, be less detrimental. The pyrite may gradually change color, become coated with a brown crust, and eventually alter entirely into hydrated iron oxide, which retains the original shape of the pyrite and thus forms a pseudomorph.

Marcasite decomposes in much the same manner, though usually more rapidly than pyrite, and more rarely in the pseudomorphic form.

SULPHIDES NOT ALWAYS INJURIOUS.

Most authors who have discussed impurities in building stone have stated unreservedly that pyrite is injurious when the stone in which it is contained is used for exterior work. This statement is not always true, however.

Many recorded examples and personal observations show that the rate of decomposition of iron sulphides is different in different deposits. In some marbles the sulphides decompose and form undesirable discolorations in a few months or a few years. Marble from other deposits may contain iron sulphide that will withstand many years of weathering, and show no appreciable change. Examples are known of American marbles containing pyrite that have stood

[a] Allen, E. T., Crenshaw, J. L., and Johnston, John, The mineral sulphides of iron: Amer. Jour. Sci., ser. 4, vol. 33, 1912, pp. 169–217.

[b] Merrill, G. P., Stones for building and decoration, 1903, p. 31.

exposed to the weather for more than 100 years without noticeable staining.

The researches of Julien and Stokes throw considérable light on the conditions governing the rate of decomposition of iron sulphides, and consequently are of great value to quarrymen and architects.

RESEARCHES OF JULIEN.

Julien[a] gives a description of pyrrhotite, marcasite, and pyrite with notes thereon that suggest that the process of metamorphism tends to alter pyrrhotite and marcasite into pyrite, and that, of the iron sulphides, pyrite predominates to a greater extent in marble than in the rocks that have undergone less metamorphism.

He points out that pyrite even in the form of crystals is rarely pure, but contains varying quantities of marcasite more or less intimately mixed with it. The author emphasizes the point that the rate of decomposition depends upon the purity of the mineral; mixtures of pyrite and marcasite weathering readily, whereas pure pyrite is stable.

The relation of specific gravity to rate of decomposition is of interest. The specific gravity of marcasite is given as 4.80, and of pyrite as 5.01. As pyrite is the heavier, any admixture of marcasite with it will lower the specific gravity. Consequently specific gravity may be taken as a criterion of purity, and therfore of resistance to weathering.

In judging the probable resistance to weathering of any iron sulphide present in a rock the author recommends careful examination of form, color, fracture, and density with supplementary tests for determining whether the mineral is pure pyrite or whether it contains marcasite mixed with it. If mixtures of the two sulphides are found, rapid alteration is to be expected. If possible, pure material, preferably crystals, should be obtained. The observations to be made and their interpretation are as follows:[b]

1. Crystal form; if isometric the material is probably resistant, though such crystals may contain marcasite.
2. Color; if brass yellow the mineral is probably nearly pure pyrite; if pale yellow or tin white marcasite predominates.
3. Fracture; if conchoidal it is probably pyrite; if uneven or granular, marcasite is abundant or the mineral is otherwise impure.
4. Hardness; the hardness of pyrite is 7 and of marcasite 6.5.
5. Odor and streak; a sulphurous odor and greenish streak indicate the presence of much marcasite.
6. Specific gravity; fairly pure pyrite can be expected only when the specimen has a specific gravity greater than 4.97.

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[a] Julien, A. A., On the variation of decomposition in the iron pyrites; its cause and its relation to density: Ann. New York Acad. Sci., vol. 3, 1883-1885, pp. 365-404; vol. 4, 1887-1889, pp. 123-223.

[b] Julien, A. A., Op. cit., pp. 221-223.

7. Observations of weathering effects in quarries or outcrops are recommended as of great practical value.

8. Tests may be made of the comparative rapidity of tarnish when the sulphides are exposed to the fumes of bromine or fuming nitric acid.

RESEARCHES OF STOKES.

Stokes [a] verifies Julien in some respects and differs from him in others. He asserts that in general marcasite oxidizes more readily than pyrite, though finely divided or porous specimens of either decompose rapidly. Mixtures of marcasite and pyrite decompose readily, probably owing partly to porosity and partly to electrochemical action between the two.

Attention is directed to the fact that the presence of various impurities, such as quartz or silicates, may lower the specific gravity of pyrite. Density alone therefore is not a sure method of detecting the presence of marcasite in pyrite unless the absence of all other impurities is known.

The author also casts some doubt on Julien's claim that most isometric crystals contain appreciable amounts of marcasite. Although exceptional specimens may contain marcasite, most isometric crystals are pure pyrite, and orthorhombic crystals pure marcasite.

METHODS OF DETERMINING STABILITY OF SULPHIDES.

Whether a low density is to be attributed to the presence of marcasite or to some other impurities is of little consequence to the stone man. The important point supported by both authors is that specimens of low density are prone to decomposition. Granular or porous specimens are likewise easily oxidized.

It would seem that by making observations and tests as suggested by Julien a fair conception could be gained of the probable stability of the sulphides present, and therefore of the adaptability of the marble for exterior structural purposes. A microscopic examination is of value as indicating whether the sulphides are present as crystals or in porous or granular form. Crystal form and specific gravity are the physical properties that seemingly give the most definite information. The most reliable information may be obtained from observing structures made of sulphide-bearing marble that have stood for many years exposed to weathering. If such observations can not be made, chemical tests may be substituted. The condition of sulphides in the quarry wall or ledge may give less reliable information. Repeated attacks of circulating ground water contain-

[a] Stokes, H. N., On pyrite and marcasite: U. S. Geol. Surv. Bull. 186, 1901, 50 pp.

ing active solvents may decompose the sulphides, although in the dry wall, or exposed only to the action of rain water, they may last indefinitely without sign of decomposition.

THE USES OF SULPHIDE-BEARING MARBLES.

Although it is true that iron sulphide is not necessarily injurious in marble, it should be carefully avoided in the selection of stone for exterior uses unless good evidence has been obtained that stains will not result. For interior structural or ornamental purposes, however, sulphide-bearing marbles may be used. In some structures the yellow bands and patches of pyrite have produced beautiful effects in polished surfaces.

SILICA.

ORIGIN OF SILICA IN MARBLE DEPOSITS.

Silica may be an original constituent of the marble mass. Marble is formed chiefly from an accumulation of calcareous remains, which have been crushed, folded, and recrystallized to a greater or less degree. It is well known that certain marine organisms, such as diatoms and some varieties of sponges, have silicious skeletons. An accumulation of such shells would form masses of silica. The occurrence of flint balls in chalk cliffs is ascribed to this cause. The silica may appear in straight or lenticular bands or knots.

Van Hise[a] claims that as most of the organic silica is in soluble form it is dissolved and later reprecipitated as chert. He emphasizes the strong tendency in minerals to segregate during the process of deposition from solution, and ascribes to this cause the occurrence of the chert in knots and lenses. The larger and more persistent bands may consist of silicious matter in its original form. The occurrence in concentrated masses may be due to the tendency of marine organisms to live in colonies.

Conditions favoring the propagation of silicious-shelled organisms probably prevailed over a wide area at the same time. If therefore silicious lenses are found in a certain bed in a marble quarry, in all probability they are characteristic of that bed over a wide area.

Silica may also be introduced into marble beds at a later stage in the history of the deposit. Water that percolates through fissures in the mass may contain small quantities of silica in solution, and this material may be precipitated in cracks and cavities. Silica in this form will tend to follow the unsoundness, and may even effectually seal up the fractures.

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[a] Van Hise, C. R., A treatise on metamorphism: U. S. Geol. Surv. Mon. 47, 1904, p. 817.

DETRIMENTAL EFFECTS OF SILICA IN MARBLE.

Silica is at least twice as hard as ordinary marble, and in consequence its presence greatly retards channeling, drilling, or sawing and is injurious to tools, especially wire saws. A flint ball may divert the saw cut to one side or may impede or entirely prevent cutting. Moreover, the unequal hardness presented by the surface of a flinty marble makes it difficult to obtain uniformity of finish under the buffer.

The presence of silica usually detracts from the appearance of marble. The flinty or cherty mass as a rule differs from the marble in color or texture and thus constitutes a blemish comparable with that produced by the presence of a knot in an otherwise uniform stick of timber.

SILICATED MARBLES.

Silicated marbles are those that contain silicates such as pyroxenes, amphiboles, mica, or chlorite. Such silicates may result from the combination of silica with the calcium or magnesium of the marble, with the escape of carbon dioxide. Marbles containing interbedded silicates may also be included under silicated marbles, although none of the silicates may have been derived from the marble.

Interbedded silicates may originate from clay beds in the original limestone. Clay brought down by rivers may be interbedded in limestone deposits formed near the shore. The process of metamorphism that changes the limestone into marble has a pronounced effect on the interbedded clay. Ordinarily it is altered into mica and chlorite. The resulting marbles will therefore contain bands of these minerals. In marbles that have undergone a limited amount of folding and deformation the mica and chlorite bands will remain conformable with the original bedding. Such bands may constitute definite bedding planes separating beds of pure marble. In such form they are not serious imperfections and may even assist the process of quarrying. If deformation has been more intense the mica and chlorite may be scattered throughout the marble mass as a dark banding. Although serviceable for certain purposes, in general a clouding or banding detracts considerably from the market value of the stone.

The formation of silicates by combination of silica with the calcium and magnesium of the marble has usually been accomplished by intense metamorphism. Dolomitic marbles may contain crystals of tremolite, a silicate of calcium and magnesium. Most of the crystals are white with a silky luster, have a characteristic diamond-shaped cross section, and are much harder than marble. They may be microscopic in size or may attain a length of 2 inches. They

break out easily and consequently are serious imperfections when of large size. White mica is a common silicate in marble. Its presence tends to make the rock more easily cleavable, as the plates are usually parallel with the grain. Wollastonite, diopside, olivine, and tourmaline are other common silicates occurring in marbles.

DOLOMITE IN MARBLE.

A marble that consists of pure dolomite is harder and more resistant to weathering than one consisting of pure calcite. Thus a marble consisting of dolomite may be no less valuable than one consisting of calcite. As there is a marked difference in solubility and hardness between the two minerals, a marble composed of alternating masses of calcite and dolomite is undesirable. If dolomite is present in lenses or bands, unequal weathering will result and produce a non-uniform surface. There is also the probability of differences in texture, color, or susceptibility to polish between the two minerals. Although pure dolomite, or intimate mixtures of dolomite and calcite, are not to be regarded as inferior types of marble, heterogeneous mixtures in the form of lenses, knots, or bands are, for the above reasons, undesirable.

UNDESIRABLE COLORS.

Marbles may have various colors or combinations of colors, many of which are attractive for decorative purposes. The colors may be permanent or they may change after exposure to sunlight or weather. The fading of an attractive color is undesirable. Vogt [a] states that certain Norwegian blue marbles fade after five years' exposure to the light. Marbles from various other localities are known to fade perceptibly after exposure to sunlight for a number of years. As marble is usually chosen for its inherent attractiveness, color is one of its chief assets, and if the color changes the rock may no longer be of value. Occasionally exposure to the weather improves the color of a marble, but as a rule permanence of original color is to be desired.

The origin of some undesirable shades of color may be easily determined. Black and gray shades are usually due to the presence of carbon. Red and brown shades may be due to manganese oxide (MnO) or to hematite (Fe_2O_3). Cream and yellow colors are caused by the presence of fine particles of hydrous iron oxide, limonite ($2Fe_2O_3.3H_2O$). If pyrite is present in a marble its appearance may be attractive when the marble is first quarried, but may give rise to stains from subsequent oxidation. Green colors may be due to sericite, chlorite, epidote, actinolite, chrome mica, diopside, or serpentine.

Certain other shades of color are of obscure origin. Some variations in color, though slight, may detract immensely from the market value

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[a] Vogt, J. H. L., Norsk marmor, Kristiania, 1897, p. 354.

of the rock. A white marble commonly shades off into a lifeless gray. This may be due to imperfect recrystallization of the gray rock, which may therefore be classed as a limestone rather than a marble. In other deposits an attractive white may fade into an inferior bluish white. Lenses and bands of the bluish material may pass irregularly through the white, and thus occasion excessive waste, or necessitate classifying the marble in a lower grade. The reason for this bluish coloration is unknown; some impurity in minute quantities or some peculiarity of crystallization may be the cause.

During the process of limestone formation there is a tendency for similar conditions to prevail over a wide area at the same time. As a consequence colors due to substances that are original constituents of the marble tend to exhibit a minimum variation in different parts of the same bed and a maximum variation in passing from one bed to another. It is usually found that certain beds in a marble deposit give more attractive colors than other beds, and that each bed exhibits more or less constancy of color. The variation from white to bluish white in different beds may be pronounced, though less marked variations have been noted in different parts of the same bed. If marble of a particularly pleasing color appears in a certain bed, there is much greater probability of finding more of the same material by following the original bed than by seeking for it in other strata.

FISSILITY.

In certain marble deposits numerous parallel cleavage planes have been developed. They may be so closely spaced that little serviceable marble can be obtained. According to Becker,[a] cleavage in rocks is due to a weakened cohesion along planes of maximum tangential strain (or maximum slide). The process of folding, which is so common in marble beds, undoubtedly causes intense strains, which may develop a cleavage or fissility in the marble. Marbles that are fissile have probably been subjected to profound metamorphism. The same author[b] points out that crystals tend to grow in the direction of least resistance, and hence mica plates, so common in easily cleavable marble, tend to grow parallel with the planes of schistosity. Thus the mica may further facilitate the cleavage. The presence of much muscovite (white mica) parallel with the cleavage planes in several fissile marbles has been noted.

Marble with a certain degree of fissility may be used if cut in slabs parallel with the cleavage. There is a probability of great waste, however, and as a rule such deposits should be avoided.

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[a] Becker, G. F., Experiments on schistosity and slaty cleavage: U. S. Geol. Survey Bull. 241, 1904, p. 11.
[b] Becker, G. F., Op. cit., p. 22.

DEFECTS IN TEXTURE AND STATE OF AGGREGATION.

Uniformity of texture is an important requisite of a marble. A noticeable variation in the size of grain from point to point detracts greatly from its appearance. Moreover, different textures decompose at different rates, and therefore nonuniform marble will tend to weather and decay in an unequal manner, and to produce a pitted and uneven surface.

Marbles vary greatly in size of grain. Most of the fine-grained marbles are more durable than the coarse grained. As pointed out by Dale,[a] acid water travels more rapidly between coarse than between fine grains. For all exterior structural purposes, therefore, fine-grained marble is to be preferred.

Aside from its durability, the fine-grained marble offers other advantages. It usually takes a better polish than coarse-grained material. Moreover it is as a rule better adapted for intricate cutting or carving, though some of the coarse-grained marbles of Georgia carve well.

The percentage of pore space is variable in marbles from different localities. In no marbles are the crystals so firmly cemented together that no pore space exists, though in some specimens it is small. Pores permit the infiltration of water, which affects the marble in two ways. Rain water may contain carbon dioxide or other solvents that hasten the decay of the rock. Water in the pores may freeze in cold climates and thus cause disintegration. Hence, low porosity is desirable in marbles for exterior uses.

However, a marble should not be condemned simply on the basis of percentage of pore space. A rock of low porosity may under certain conditions decompose more readily than one of much higher porosity but having a different type of pores, as explained in the following paragraph:

The rate of solution of marble by circulating water depends on the nature of the solution and the rate of circulation. The nature of the solution depends on environment, and not on any inherent peculiarities of the marble itself, except in so far as the solution is modified by substances dissolved during its passage through the rock. The rate of circulation, however, depends largely on the nature of the pores. If the pores are isolated from each other by walls of rock, circulation must be slow. If, on the other hand, they are connected by open channels, free and rapid circulation is possible. The rate of circulation depends, therefore, rather on permeability than on percentage of porosity. The permeability of marble may be tested by investigating the rate of flow of water forced through it under pres-

[a] Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Surv. Bull. 521, 1912, p. 37.

sure as described by Parks,[a] or it may be tested by the distance to which colored analine dyes will pass into the rock in a given time. Naturally those marbles of high permeability are undesirable for exterior work.

In cold climates the effect of frost on exposed marble may materially increase the rate of decay. Although low porosity undoubtedly tends to diminish the danger of deterioration through frost action, the effects of frost are not proportional to the percentage of pore space. As shown by Buckley,[b] the effect of frost on a stone depends on the rapidity with which the stone gives up its included water. Parks[c] in commenting on this statement brings out the following facts: The injury is caused by the expansion resulting when water changes into ice. If the pores are full of water, expansion must cause disintegration of the rock, but if the pores are only partly full, expansion may take place without rupture of the pore walls. Capillarity may keep many of the finer pores full of water, whereas air in the larger cavities prevents complete filling. It is evident, therefore, that a stone having a large proportion of fine pores is in greater danger from frost than one in which the proportion of fine pores is lower. In cold climates, therefore, numerous fine pores are undesirable in marble that is to be exposed to the weather.

Porous stones readily collect dust particles and therefore become dirty much more rapidly than compact varieties. Porous stone should therefore never be used if it is to be exposed to excessive smoke or dust.

In sawing porous marbles sand must be used as abrasive. Although crushed steel cuts more rapidly, it may enter the pores, and later scratch the slab during the process of polishing, or may by rusting cause serious stains.

The strength of a marble depends in part on the state of aggregation of its constituent particles. Marbles in which the particles adhere to one another with great tenacity are stronger and more durable under pressure than those in which the grains are loosely coherent. A remarkable interlocking of crystals has been noted in some marbles, and it is thought that such a condition increases their crushing strength.

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[a] Parks, W. A., Report on the building and ornamental stones of Canada: Can. Dept. Mines, vol. 1, 1912, p. 61.

[b] Buckley, E. R., Building and ornamental stones of Wisconsin: Wis. Surv. Bull., 4, 1898, p. 22.

[c] Parks, W. A., Op. cit., pp. 63, 64.

PROSPECTING AND DEVELOPING MARBLE DEPOSITS.

PROSPECTING.

VALUE OF GEOLOGIC MAPS.

Most marble beds outcrop in long and narrow bands, which may extend for many miles. These bands represent truncated edges of folds in the rock and may be curved or straight, depending upon the topography and on the nature of the fold. Much of the rock surface may be covered with gravel, sand, or clay to a considerable depth. The geologist may, by a careful study of outcrops exposed here and there, obtain a knowledge of the chief structural features, and may thus determine the position, thickness, and attitude of the marble beds with a fair degree of accuracy, even if they are entirely hidden by surface débris. If geologic maps of marble belts are carefully made they are of inestimable value to the marble prospector. By accurately locating himself in the field and carefully consulting a geologic map the prospector may determine the position of marble belts beneath the soil and know something of their extent and attitude although they are unseen. It is important, therefore, that all available geologic maps of the region be consulted freely.

NEED OF DETAILED PROSPECTING.

Knowledge of the suitability of any particular site can be gained only by detailed prospecting including a determination of the depth of overburden and of surface alteration of the rock and of the extent, quality, impurities, and soundness of the deposit. It is extremely unwise to proceed with development work without a reasonable assurance that an available mass of sound and attractive marble is sufficiently uniform in quality and abundant in supply for profitable exploitation.

DETERMINATION OF OVERBURDEN.

The depth of stripping necessary may be determined at small cost by putting down drill holes. The need of such preliminary tests should be recognized. In certain instances stripping has been attempted without any previous investigation of the depth of soil to be removed. The great loss that may result from thus working blindly may be illustrated by one particular instance. A pit opened by a certain quarry company and later abandoned is 60 by 80 feet in extent and 20 feet deep. To make such a pit required the removal of approximately 3,550 cubic yards of soil, which, at an average cost of 25 cents per cubic yard, would have cost $887. The only purpose this pit actually served was to show the owners that the stripping was too deep to make quarrying at this point profitable. The same informa-

tion could have been obtained by projecting two drill holes each 25 feet deep. At the ordinary cost of rock drilling, $2 per foot, the cost would have been only $100, a saving of $787.

In estimating the necessary cost of stripping for a new quarry the attitude of the marble beds must be taken into account. If the beds are flat a greater area of rock must be uncovered than if they are steeply inclined or vertical.

Naturally conditions relating to disposal of stripping are of great importance. In certain places it is a matter of some difficulty to find a suitable place in which to deposit the soil that must be removed; in other places the soil may be carried to neighboring valleys or low-lying areas.

SURFACE STUDY INSUFFICIENT.

Surface observations are of great value, especially as regards jointing. The process of weathering tends to emphasize all unsoundness and thus facilitates the study of joint systems. Exposed surfaces may also permit a determination of dip and strike and the thickness of the beds.

In determining the quality of a marble deposit a study of uncovered knobs or ledges should not, however, be deemed sufficient. On account of surface weathering the top rock may differ materially from the deeper parts of the deposit. Moreover, the number and spacing of joints at the surface may be no indication of the prevailing conditions at depth. Before the prospective quarry operator installs expensive derricks and hoisting machinery and purchases channeling machines and drills he should have a fair idea of the quality and soundness of the marble and the supply available. In order to obtain this knowledge drill cores should be taken at several points.

DIAMOND-DRILL PROSPECTING.

The ordinary diamond drill will give the necessary information regarding color, uniformity, and general appearance of the stone, and also the extent of the formation. It will not, however, give definite information concerning the dip and the strike or the unsoundness of the marble. If drill cores come out in long, unbroken sections which show no indication of cracks, it may be assumed that the rock is fairly sound. If, on the other hand, the core is in short sections, the rotation of the drill will in most cases have so worn and rounded the broken ends that it will be impossible to determine whether the breaks are due to natural planes of weakness in the rock or to the process of drilling itself.

MECHANISM OF THE DOUBLE CORE-BARREL DRILL.

A method of prospect drilling that has been employed by at least one operator involves the use of the double core-barrel drill that was designed primarily for drilling bituminous coal, and operates in such a manner as to bring out a core from a delicate material with a minimum of breaking or other damage. The drill, shown in Plate II, consists of an outer and an inner tube. The outer is attached to the drill rod by means of the core-barrel plug, and is rotated the same as the cutting tools. The inner tube is suspended by a ball-bearing plug at the top and centered by a ball bearing near the bottom. Thus, while the outer tube revolves, the inner tube which carries the core ring or lifter at its lower end remains stationary. The core passes up through the bit into the inner barrel, where it is protected from friction from the rotating parts, and from the washing action of the water. An ample water supply is provided between the two tubes. The purpose of the inner tube is to hold the core as immovable as possible, and thus to prevent the wearing or rubbing of the broken ends. Although the core does not revolve after it passes through the core lifter into the inner tube, the principle of cutting and the action of the bit on the core until after it passes into the inner tube are just the same as with the single-barrel drill. Therefore, if a natural parting is encountered in the rock, or if for any reason the core is broken off, it is liable to be rotated somewhat before it gets into the inner tube. If the core breaks obliquely the danger of rotation is minimized. Cores should be at least 2½ inches in diameter.

VALUE OF DOUBLE CORE-BARREL DRILL TO PROSPECTOR.

The use of a drill like that described enables the prospector to judge the unsoundness of the marble at points beneath the surface. If one examines carefully the ends of the sections of drill core one can, almost without exception, interpret the breaks, and state whether they are due to natural joint planes in the rock or to the process of drilling. If the cores are properly oriented, the proximity and direction of all natural cracks in the rock in the immediate vicinity of the drill holes may thus be ascertained.

If the marble deposit is well exposed, the dip and the strike may be determined from examination of the outcrops. If, however, it is completely buried, these features may be determined from the drill cores if they are carefully oriented.

DRILLING THE HOLES.

NUMEROUS HOLES UNDESIRABLE.

Information should be obtained with a minimum number of drill holes. In this respect prospecting for marble differs materially from prospecting for metalliferous ores. As regards metalliferous ore, the soundness of the ore is not important, whereas with the marble every crack or cavity increases the proportion of waste in the quarried product. A drill hole in a quarry is quite as objectionable as a crack. If the deposit lies flat or nearly so, a single well-placed core driven entirely through the deposit will give information as to the character of the marble, and show whether it is one homogeneous mass, or is divided by streaks of color or open beds into different layers, and whether the layers differ in character one from the other.

PROSPECTING THICK BEDS DIPPING AT A MODERATE ANGLE.

If the deposit dips at a moderate angle and is comparatively thick, the best way to determine its thickness and the character of its beds is to lay out a line of drill holes at right angles to the strike. The first drill hole that penetrates the upper beds should begin in the hanging wall, a name given to the bed immediately overlying the marble band. The holes should be of such depth and spacing that the bottom of a hole in the upper beds will penetrate the same layer as the top of the neighboring hole on the side toward the footwall. The core nearest the footwall should reach and penetrate this wall. By this method a series of core holes of moderate depth will supply samples from all the beds, and the relatively high cost of drilling deep holes will be avoided. The same information could be obtained by drilling one hole, starting at the hanging wall and penetrating the entire deposit to the footwall, but the cost would probably be much higher.

DETERMINATION OF DIP AND STRIKE FROM DRILL CORES.

If the marble has at any time been subjected to severe stresses, and has been sufficiently plastic to flow freely under the pressure, the original distribution of color may be so disturbed that it no longer bears any well-defined relation to the bedding. Ordinarily, however, the direction of veining or clouding is approximately parallel with the bedding. A vertical core may therefore enable the prospector to determine the average angle of dip of the marble if he measures the angle at which the streaks of color or the grain of the rock traverses the core itself.

If access to the surface of the marble can be obtained before the core is started, the direction of strike may also be determined from a single vertical core by orienting the upper part of the core as it was

DOUBLE CORE-BARREL DRILL.

before drilling, and then by carefully matching the successive pieces. If the surface is inaccessible three cores distributed on the corners of a triangle may serve to determine both dip and strike, provided there are well-defined beds that may be matched in the different cores. If a sufficient area of the marble is stripped the strike and dip can be determined by direct observation, but even then the indications of the core may be of value, because the strike and the dip may vary considerably even within short distances.

ARRANGEMENT OF HOLES TO TEST VARIATION IN QUALITY.

It may seem desirable to test the uniformity in quality of marble in the same bed at different points along the strike. To get a fair idea of any variation in quality it is well to lay out, across the strike, two parallel lines of holes, 300 to 1,000 feet apart, depending on conditions. Every effort should be made to locate the two lines of holes so that each core in the second line will come from the same part of the deposit as the corresponding core of the first line. Thus, fair conclusions can be drawn as to the variation in quality of any layer of the deposit as a whole from point to point. All drill cores should be polished on one side in order to facilitate determination of color, uniformity, and degree of polish that may be obtained.

As core drilling is rather costly, it is well to supplement the evidence of the cores by stripping the marble along each line of holes. It is wise also to dig an occasional trench at right angles to each line of core holes so as to expose the marble to some extent along the strike.

ARRANGEMENT OF HOLES TO DETERMINE UNSOUNDNESS.

A marble deposit in which the color, texture, or other qualities are highly satisfactory may nevertheless not warrant commercial development because of joints. Most joints occur in two systems, the cracks in each system being approximately parallel with each other, and the two systems more or less at right angles. Occasionally more than two systems are present. The spacing of the cracks is subject to wide variation in different deposits and even in different parts of the same deposit. Near the surface the cracks are usually more numerous and more irregular than at depth. Nearly always at least two systems of cracks will persist with more or less prominence to almost any depth to which quarrying operations may be economically carried. It is important to determine as early as possible which of the cracks that appear at the surface are likely to persist and also their nature and spacing in the deeper parts of the deposit.

Most of the cracks are nearly vertical and also nearly at right angles to the bedding of the marble, few cracks deviating from a

right angle more than 30°. Hence, a vertical core taken out of marble that is rather unsound may reveal the presence of few of the cracks. Therefore, a vertical hole is not reliable as a means of estimating the unsoundness to be encountered.

Having determined by any means available the systems of cracks that will probably persist at depth, the general compass bearing of each system should be determined. A series of core holes approximately parallel with the bedding should be laid out, the compass bearing of the core hole being such that it will intersect both systems of cracks at approximately the same angle. A core projected in such a direction will give a fair estimate of the unsoundness present. It also results in an oblique break in the core wherever it crosses a crack of either system, and thus makes less likely any grinding of the ends of the core sections. The origin of a fracture may be determined with reasonable certainty, because a freshly broken surface made in sound marble differs materially in its appearance from the surface of a break due to a joint.

It is important to take cores from near the top, from near the middle, and from near the bottom of the deposit, because the unsoundness often varies in different beds as well as in different parts of the same bed.

If the marble deposit stands at a high angle, one set of core holes driven in an inclined direction and penetrating from the hanging wall to the footwall, or the reverse, can be laid out so as to give the information required as to the quality of the stone and also the unsoundness.

It is practically impossible to take out good cores that are representative of the deposit from horizontal drill holes. The core from a horizontal hole invariably breaks into short pieces, which grind on each other, in spite of the use of the double-core barrel. The core hole has to be slightly larger than the outer diameter of the core barrel in order to permit the passage of water and slush and also to prevent binding of the drill in the hole. As a result the progress of the drill brings a considerable weight on the unbroken end of the core, and as soon as the core is a few inches long it breaks off. The break occurs seemingly at the bottom of the hole, the result being that the spring clips on the inner core barrel, which are supposed to hold the unbroken end of the core and thereby prevent rotation of the inner core barrel, have nothing on which to hold. During the time that the drill is cutting a little farther in and the clips are getting a fresh hold, the inner core barrel rotates to a greater or less extent, so that the core ends grind on each other. Therefore, if the marble beds lie flat or nearly so, unsoundness must be prospected for by inclined core holes; otherwise the cores will not yield the information desired.

ARRANGEMENT OF THE CORES OBTAINED.

In order to get the fullest information from an inclined core hole the core parts should be matched up from one end to the other, and placed, a part at a time, on an inclined rack that will hold the core in a position parallel with the hole from which it was taken. While the core is in this position the compass bearing of the cracks and also the angle that they make with the core can easily be determined. From this information a plan may be made from which the probable percentage of marble unaffected by unsoundness may be computed with reasonable accuracy.

Before an inclined core is started, the first piece should be marked so that it can be laid top side up after it has been removed. The dip of the core hole should be slightly different from the dip of the beds so that a streak of color, the grain of the marble, or any other feature parallel with the bedding will traverse the core at a slightly oblique angle. With such a method of drilling, if clean right-angled breaks occur, one piece of core may easily be matched to the preceding piece, and the prospector may be sure that he has the entire core properly matched from one end to the other.

VALUE OF THE INFORMATION GAINED FROM CORES.

From a careful study of cores taken out as described above, valuable information can be obtained in reference to the unsoundness existing at some depth in the deposit. It will often happen that, although the marble appears so broken at the surface that profitable operation seems impossible, investigation with a drill in the manner above described will reveal the fact that the cracks close and disappear to such an extent that profitable operation may be possible after the top rock has been removed.

A later section discusses the advantages of quarrying in accordance with unsoundness, and shows that a knowledge of the extent, direction, and character of the unsoundness is important in laying out the preliminary opening in order to avoid unnecessary expense.

COST OF MARBLE PROSPECTING.

Prospecting as described above is rather costly. The average cost of double core-barrel drilling by contract is about $3 per lineal foot. If a marble company has sufficient work to justify owning its own drills the cost can be considerably reduced. In any case the information desired can not be obtained for less than $5,000 to $20,000. Many marble deposits, however, can not be profitably worked except on a large scale, so that a considerable outlay made to determine whether conditions are favorable is money well expended.

PRESERVATION OF CORES.

As a rule, drill cores are not preserved with sufficient care by quarrymen. They are often carelessly stored, lost, or given away as samples. It is important that every part of every drill core be carefully marked and stored for future reference. It must not be assumed that the value of drill cores disappears after their first investigation. They are invaluable records which should be available at all times.

STRIPPING.

ORDINARY METHODS OF STRIPPING.

Most marble quarried is of a high grade, and the bulk of rock removed is small in comparison with the quantity handled in many quarries where rock is obtained for cement or road construction. As a consequence the area stripped is usually not great enough to justify the use of steam shovels.

When the surface of a deposit is fairly level, teams and scrapers may be used to advantage. Usually, however, the overburden of soil is so placed that it must be removed by means of large pans which are loaded by hand, and handled with a derrick hoist. In order to remove the soil to a sufficient distance from the excavation, cars and tracks may be necessary. A serviceable car for this purpose is shown in Plate III, *A*. The loaded pan is placed on the car by means of the quarry derrick. Horizontal iron bars on the undersurface of the pan are placed in the iron sockets of the car. These trunnions are so placed that a little more than half the weight of the load is toward the rear end of the car. When the car reaches its destination, the back of the pan is raised, thus overbalancing the load, and dumping it from the front of the car. Where soil must be removed to a great distance, the loaded pan may be dumped into railroad or cable cars.

INSUFFICIENT REMOVAL OF STRIPPING.

A common mistake in the process of stripping is to remove the soil and waste to an insufficient distance from the excavation. The desire to attain quick results at small expense, and lack of foresight regarding the probable extent of future operations are the chief causes of insufficient removal. As a consequence, quarrymen may find after a few years' operation that they must handle material a second time, thus adding greatly to the expense of quarrying. The extensive marble workings of Italy, in the neighborhood of Carrara, Massa, and Seravezza, are greatly hampered by accumulations of rubbish which have buried vast quantities of good marble. The market price is too low to allow the removal of this débris.

A. SERVICEABLE CAR FOR REMOVAL OF WASTE AND STRIPPING.

B. TYPICAL EXAMPLE OF STEAM LEAKAGE IN A QUARRY TRANSMISSION LINE

C. TRACK SUPPORTED IN LEVEL POSITION FOR "SAW-TOOTH" FLOOR CHANNELING.

Lack of foresight is also shown at some quarries where quarrymen dump their waste material into abandoned excavations. This procedure may be justifiable if there is no probability that the pit may ever be reopened. The practice has been observed, however, in places where it seemed that greater success would have been attained by going deeper in the old pits than by making new ones.

WASTE HEAPS IMPEDING FUTURE DEVELOPMENT.

Another mistake in the disposition of waste material is due to the inability of the operator to foresee the direction in which future operations will extend his quarry. As regards steeply tilted beds, it is obvious that if excessive depth is to be avoided development can take place only in the direction of strike. Nevertheless, at certain quarries of this type observed, stripping has been deposited directly in line with the strike of the beds, and thus an extension of the quarry excavation must soon overtake a great heap of accumulated débris.

USE OF OVERHEAD CABLEWAY HOISTS.

For the removal of both soil and waste rock to a sufficient distance to avoid interference with future operations an overhead cableway hoist may be desirable. At one quarry at which such a cableway is used the hoist engine is mounted on a truck that travels on a curved track. It can thus be shifted to strip in different places. Where difficulties are in the way of removal of stripping by derrick or car, the advisability of adopting a cableway system may be considered.

HYDRAULIC STRIPPING.

Hydraulic stripping is employed with success in several marble quarries. There are certain conditions, however, that must be met in order that hydraulic stripping may be successful or even possible. Two important conditions are an adequate water supply and easy drainage. If the water supply is obtained from drilled wells or small streams that may go dry, the process will probably fail.

The soil removed by hydraulic stripping may be disposed of in several ways. It may be carried away in a stream valley and deposited naturally at various points along the course of the stream. In some places, however, the deposition of the soil along the stream valley would be detrimental to agriculture or to other interests. In that event, a dam may be built forming a settling basin. In certain operations, surface soil is conveyed to abandoned quarry pits. With such disposal surface drainage is also necessary in order to get rid of accumulated water. Also one must be sure that there will be no future desirability of reopening the pit, as reopening would not be feasible after the pit was once filled with soil.

In certain quarries in Georgia, the good marble is underlaid by a hornblende rock. When in the process of quarrying the hornblende rock is reached, the pit is abandoned, and can then be used as a settling basin in the process of hydraulic stripping.

Those who have had experience in hydraulic stripping estimate that the cost of soil removal by that method may be as low as 2 cents per cubic yard.

GENERAL PLAN OF QUARRYING.

PRODUCTION OF UNIFORM GRADES OF MARBLE DESIRABLE.

Success in marble quarrying nécessitates an adequate supply of uniform material. The inherent qualities of certain marbles win immediate popularity. With other types it has been found that by profusely scattering samples and following them with structures in various localities, public taste may be educated to demand a certain type of marble. It is most discouraging for a quarryman who has won popular favor to find that he can no longer match his samples because of failure of the deposit. The desirable bed may have pinched out, or insufficient production may be due to the operator's inability to foresee the general plan of quarry development that ought to have been followed. If the desirable bed lies flat and near the surface, extensive stripping is necessary. If the bed is inclined and the overlying beds are too unsound to permit tunneling, quarrying must follow along the strike of the beds, also demanding extensive stripping. If in either instance the cost of stripping is excessive, the quarrying of the bed must be abandoned.

FACTORS GOVERNING PLAN OF DEVELOPMENT.

The most successful mode of operation can not always be foreseen. Certain geologic factors, however, may give information pointing to the most logical method of developing the deposit. Three chief factors are the attitude of the beds, the depth of overburden, and the uniformity of the product in a given bed and in successive beds, and these three factors are intimately related. Before operations are begun the intelligent prospector will determine, in a general way at least, the necessary depth of stripping, the dip, and the strike of the beds, and the uniformity of the marble, so that he may plan a logical quarry development. The influence of the various factors on the plan of operation is discussed in the following paragraphs.

EFFECT OF ATTITUDE OF BEDS ON PLAN OF QUARRYING.

Most marble beds are situated in regions of extreme folding, and most of the beds, although originally flat, are tilted, though a few are level or nearly so. The attitude of the beds is of great importance to

the quarryman, especially if the various beds differ from each other in texture or color. If beds are inclined at a moderate angle, either long shallow quarries or tunnels must be made in order to keep up the supply of stone from a particular bed. If the beds are steeply inclined or vertical, either long, shallow quarries involving a great area of stripping, or deep quarries with their associated dangers and expense are necessary. If the strata are flat and the desirable bed is near the surface, a wide, shallow quarry results.

It is clear that if beds lack in uniformity, their attitude has a direct bearing on the most desirable plan of quarrying. If, however, the bedding is indistinct and a number of contiguous beds are uniform in color and texture, as in certain quarries in the Knoxville region of Tennessee, the attitude of the beds has a minor influence on the type of quarrying to be employed.

EFFECT OF OVERBURDEN ON PLAN OF QUARRYING.

If the marble lies in approximately horizontal beds of limited thickness, the production of large quantities will necessitate the stripping of a wide area. If the overburden of soil or superficial waste rock is great, the cost of stripping may absorb all profit. If a sufficiently strong roof is available, extensive stripping may be avoided by employing tunnel methods.

As regards flat-lying, uniform beds of great thickness, a heavy overburden will tend to promote deep quarrying, whereas a light overburden will encourage the development of wider and shallower pits.

If the beds are vertical or steeply inclined, a heavy overburden will result in deep quarrying or tunneling, whereas if only light stripping is necessary a greater lateral development in the direction of the strike is possible.

EFFECT OF UNIFORMITY OF PRODUCT ON PLAN OF QUARRYING.

In a few deposits thick beds are uniform throughout. However, in many marble regions a certain bed supplies stone of better quality than the beds above or below it. Obviously it is desirable to develop the quarry in such a manner as to obtain a maximum supply of the high-grade material. The shape of the opening, whether it shall be deep or shallow, open pit or tunnel, depends chiefly on the attitude of the beds and the depth of stripping required. Thus one may see how intimately the three factors of attitude, overburden, and uniformity are related, and how necessary it is that they be understood before development is attempted.

QUARRY OPERATIONS AND EQUIPMENT.

POWER PLANTS.

TYPES EMPLOYED.

For a discussion of various power plants and their relative advantages and economies the reader is referred to the work of Brunton and Davis.[a] On account of the special types of machinery employed in marble quarrying, some additional notes bearing particularly on this subject are given herein.

The following types of power plants have been observed in marble quarries in the United States: (1) Steam only; (2) steam for channelers and tripod drills, with auxiliary air compressor for hand drills; (3) compressed air generated by steam and conducted to the quarry by pipe line; (4) electricity developed by steam; (5) compressed air developed by purchased electric power and transmitted to the quarry by pipe line; (6) purchased electricity transmitted directly to the quarry; (7) hydroelectric power plants owned and operated by the quarry company, the electricity being transmitted directly to quarry machinery.

ADVANTAGES OF ELECTRICITY.

The superiority of electricity over any other form of power for quarry operation has lately been demonstrated. This superiority is due chiefly to the recent development of electrically operated machinery. The electric air channeler which is now in common use consumes on an average only 10 horsepower for the machine itself, although a 15-horsepower motor is required to drive it. A steam or air channeler requires a steam capacity in the boiler that, if applied to an economical generating unit, would develop at least 50 to 60 horsepower. The difference in power consumption between the electrical and other machines is due partly to differences in transmission or transformation losses, and partly to the higher efficiency of electrical machinery generally. The electric air channeler is, therefore, much more economical of power for the same results than the other types.

If the nature of the rock is such that "jackhamers"[b] are used in lieu of other types for quarry drilling, an air compressor is necessary. Should a direct-acting electric drill be produced that can do the work of a jackhammer, compressed-air transmission lines might be eliminated from the quarry. Such a drill has not as yet been developed, at least not in a practicable form.

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[a] Brunton, D. W., and Davis, J. A., Safety and efficiency in mine tunneling: Bull. 57, Bureau of Mines, 1914, 271 pp., 6 pls., 45 figs.

[b] Term applied by the manufacturer to a nonreciprocating rock drill, worked without a tripod, and provided with an automatic rotating device. The word is also spelled "jackhammer" in mining literature. It uses hollow steel through which the exhaust air passes and blows the cuttings from the drill hole.

Wire is a more convenient means of transmitting power than pipes or hose. Steam requires pipes with flexible joints for operating channeling machines. Steam pipes are often in the way and there is considerable loss of time in adjusting them in proper position. Compressed air may be transmitted through flexible hose, which is more easily adjusted than steam pipes, but transmission by wire is more convenient than either. If quarry operations are carried on in tunnels, electricity is especially desirable. A network of pipes is always a great hindrance to quarry operations.

Moreover, with electrical operation the percentage of loss in transmission can be determined and kept within small limits. With either steam or air there is a continual struggle with leaks, and radiation and transmission losses are usually excessive.

SOURCE OF ELECTRIC POWER.

If quarry operations are extensive and sufficient water power is within easy distance, it is probable that a hydroelectric plant owned and operated by the company is best, although the first cost may be high, especially if large dams must be built.

Uniformity of water supply is an important factor. If, during certain seasons, the water supply diminishes and it thus becomes necessary to construct auxiliary steam plants or to shut down operations for a part of the season, a hydroelectric plant is of doubtful advantage. Moderate variation in supply may be overcome by constructing a large reservoir.

In central Vermont three hydroelectric power plants supply most of the power required. Auxiliary steam plants supply additional power in times of low water. Transmission lines pass along the great marble valley for a distance of 62 miles. Power from various supply stations is turned into these lines and is drawn off at the quarries and mills scattered up and down its length. The power line is comparable with a great reservoir, into which water is pumped, and from which it is drawn off by pipe lines to supply various needs.

A Colorado marble company operates a hydroelectric plant with a 380-foot head of water. Pipes are laid up the mountain side and intercept streams at a high level. Consequently only small dams are required. Under such conditions a power plant can be established with a relatively small outlay of capital.

If electrical power can be obtained from a power company operating large central stations at a cost not exceeding 1½ cents per kilowatt-hour, it is probably better to purchase power than to erect a power plant at the quarry. If there are no available power lines, or if electrical power costs on the average more than 1½ cents per kilowatt-hour, and if the power required is 300 horsepower or

more, it will probably be more economical to erect a steam-driven generating plant at the quarry, provided the engine is of the condensing type. With coal at not more than $3 per ton such a plant can be run economically.

At one quarry where the installation is of 500-kilowatt capacity with an average maximum load of 400 kilowatts, diminishing at night to 200, the total cost of power delivered on the main bus bars of the generating station is less than 1 cent per kilowatt-hour. Coal at this point costs a little over $2 a ton.

The important point is that new quarries should adopt electrical operation from the outset if its adoption will be feasible at any stage of development. Current should be obtained in the most economical manner possible. Determination of the proper source of power often requires the advice of a consulting electrical and mechanical engineer.

COMPRESSED-AIR AND STEAM EQUIPMENT.

If for any reason electrical operation is inadvisable, operation by compressed air is unquestionably more economical than by steam, and vastly more convenient.

Direct steam power is undesirable on account of the transmission losses and the obstruction occasioned by the necessary network of pipes along the quarry floor. If steam must be adopted, care should be taken to reduce to a minimum the losses due to leakage and radiation.

As steam pipes are subject to varying temperatures and pressures, constant attention is necessary to avoid leakage. Plate III, *B*, shows a typical example of waste by leakage in a quarry transmission line. Many quarrymen fail to realize that money is pouring out of these leak holes at an incredibly rapid rate. A good pipe fitter should be on hand to remedy such conditions immediately.

Serious losses are also caused by radiation of heat from the steam pipes and by energy used to overcome the friction of the steam against the inner surface of the pipe. Such waste of energy may be minimized by properly covering the pipes with heat-insulating material. A few quarrymen have insulated their transmission pipes, but by far the greater majority have, as yet, made no attempt to do so. An additional means of reducing loss by condensation is the use of a superheater. Steam will not condense until it has lost its superheat, and consequently condensation of steam in pipes will not occur as long as any of the superheat remains. One degree of superheat is commonly conceded to carry steam 10 feet, but of course the net result depends on the diameter of the pipe and the condition of the insulation. If the factor mentioned be assumed, it may be con-

cluded that if steam is to be conveyed 1,000 feet 100 degrees should be added to its temperature in order that no condensation shall take place. In order to avoid as much as possible of this loss by condensation, the power plant should be situated close to the excavation.

If quarrymen could see the combined loss by leakage and radiation for a given period converted into a pile of coal they would realize that the loss assumes alarming proportions. One southern quarry company, using a battery of four 300-horsepower boilers, found by practical test that on a day of average temperature the full energy developed by one boiler was required to maintain the necessary steam pressure when no work was being done.

In old installations now operating with steam or compressed air any changes or additions ought to be determined with a view to the ultimate electrification of the entire plant. Neglect of this precaution may finally bring the operating company to a position where it can not compete with other companies that have electrified their plants.

CHANNELING.

CHANNELING MACHINES.

Sullivan, Ingersoll-Rand, and Wardwell channeling machines are all used, and each type has its special advocates. For steam operation the Sullivan machine seems to be the favorite, whereas the Ingersoll electric air type is the favorite in quarries having direct electric connection.

The most suitable Sullivan machine for marble quarrying is probably the double-swivel channeler which can be used for straight vertical cuts, for undercutting, or for cutting out corners. A few quarries, in which operations are scattered over a wide area and in which electricity is not used, employ Sullivan machines with portable boilers attached. One Georgia company operates Sullivan "duplex" channelers, consisting of two machines on a single truck working in the same channel, as illustrated in Plate IV. For long cuts such channelers are undoubtedly advantageous, as one operator can manage two machines. For small quarries where the cuts are short and many corners must be cut one machine would probably be idle so great a share of the time that the advantage would be doubtful. These machines are commonly termed "double headers" to distinguish them from the Wardwell "duplex" channelers.

The electric air channeler is self-contained, all the mechanism being on the channeler truck. The air is compressed by a motor-driven "pulsator." The air is never exhausted into the open but is simply driven back and forth under pressure in a closed circuit. The machine may be used for vertical, inclined, or horizontal channeling. The roll guides between which the channel bits run are effective for

horizontal or inclined cuts. It is better balanced than most machines for cutting along the lower side of an inclined track.

It is claimed by some quarrymen that the electric air channeler will cut at least 10 per cent more than steam or air machines if all are equally well handled. This superiority is not due to the fact that the electric air channeler cuts faster while it is cutting, but there are fewer and shorter interruptions for adjustment, setting up, etc., so that a higher time efficiency is obtainable. Cables and wires are more quickly adjusted, are less in the way, and are more easily handled than steam pipes or hose. Also, many quarrymen think the electric air channeler is the most economical in power consumption of any machine yet devised.

The Wardwell "duplex" channeler cuts two parallel channels at the same time. On a level floor and with sound stock it gives good service. As pointed out later, in quarrying unsound stock it is sometimes desirable to vary the spacing of the channel cuts in order to make them coincide with joints. With the Wardwell machine such variation would not be possible, and in consequence it is not well suited for working in unsound marble. One company has successfully adapted this machine to electric operation.

Experiments in one company's quarries in North Carolina indicate that in channeling hard marble, light and rapid blows are more effective than slower and heavier ones. The change from heavy to light blows can be made in most machines by changing the stroke.

CHANNELING IN RELATION TO BEDDING.

INFLUENCE OF ATTITUDE OF MARBLE BEDS ON CHANNELING.

Most marble deposits occur in regions where great folding and contortion of the rock has taken place as a result of tremendous geologic forces. The original flat-lying beds may be tilted at all angles. This condition greatly complicates the process of quarrying. In numerous instances the beds are separated by open fissures which demand a process of cutting in conformity with them. In other quarries the beds may exhibit no planes of separation. A distribution of color or impurities parallel with the bedding may, however, constitute factors of equal importance in their influence upon the plan of rock removal.

If conditions are at all favorable, it is desirable to maintain a level quarry floor. Conditions may be such, however, that efficiency in quarry operation or reduction in the proportion of waste may demand that quarrying be conducted on an inclined floor. No absolute rules can be given, for, as pointed out later, the nature of the product and the uses for which it is to be employed have a direct influence on the most feasible quarry method.

SULLIVAN "DUPLEX" CHANNELER.

LEVEL-FLOOR CHANNELING.

Figure 5, A, shows uniform rock with no open bedding planes and no decided rift. The beds may be horizontal, vertical, or inclined. This condition prevails in some of the Tennessee quarries. The bedding is so indistinct that it can be recognized only by the characteristic "crow feet," as otherwise the rock is uniform throughout. Under such favorable conditions, channeling may be carried on with a level floor, whatever the attitude of the beds may be.

Figure 5, B, shows marble with open bedding planes, the beds being either horizontal or vertical. In either case a level quarry floor may be maintained.

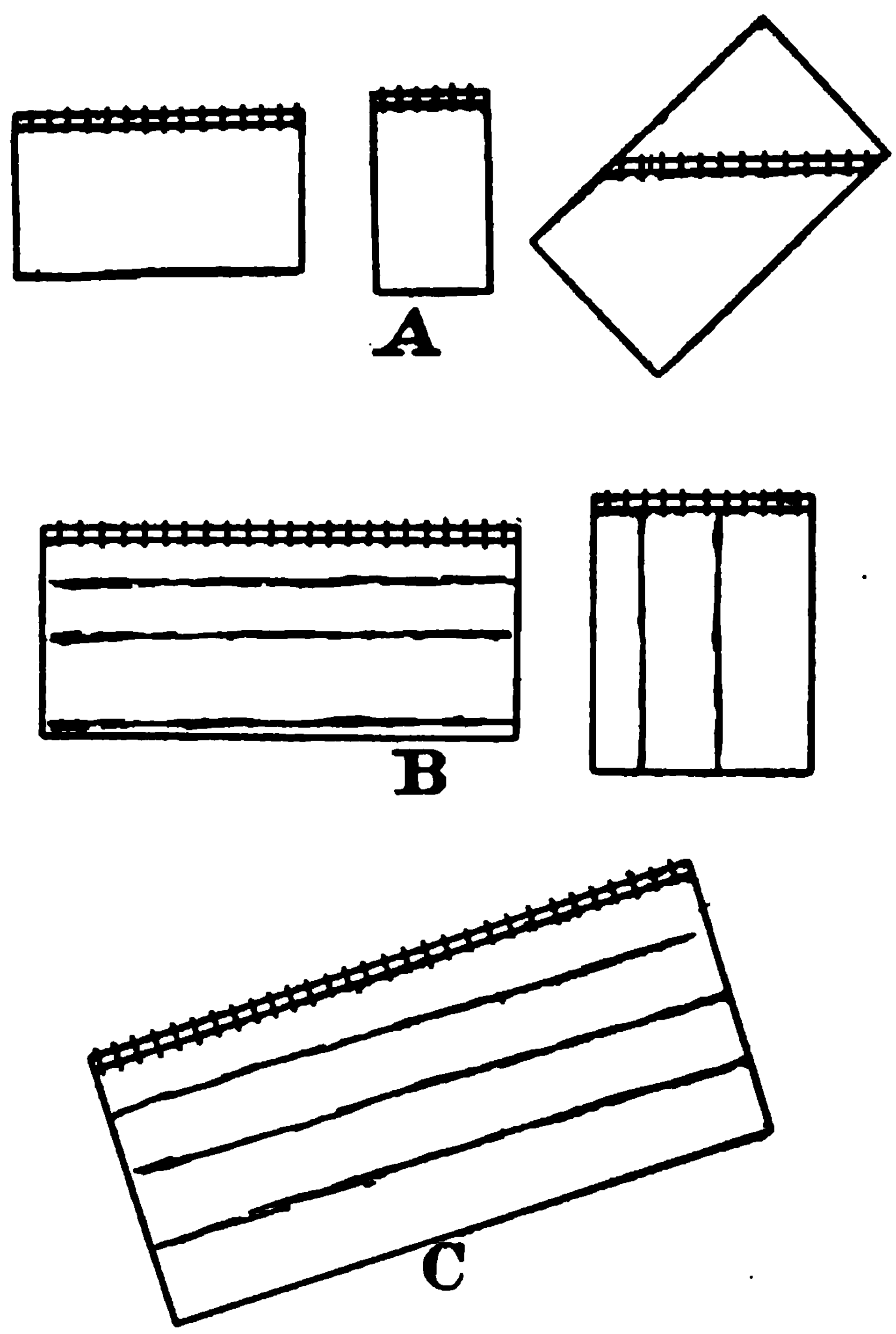

FIGURE 5.—Proper positions of channeling machine tracks for beds of different inclinations. A, horizontal, vertical, or inclined beds that have no open bedding planes or decided rift; B, horizontal or vertical beds separated by open bedding planes; C, open beds inclined at an angle less than 45°.

SAW-TOOTH FLOOR CHANNELING.

Figure 5, C, illustrates open marble beds, inclined at an angle less than 45°, such as are worked in a number of Tennessee, Alabama, and Vermont quarries. It is customary in some of the quarries to excavate right-angled blocks and to remove a row of blocks from each successive bed, resulting in the formation of a saw-tooth quarry floor as illustrated in figure 6.

In quarrying a saw-tooth floor the channeling-machine track is supported by timbers as shown in Plate III, *C*. The disadvantages of such a method are well known to many quarrymen, but are accepted as inevitable because no better method has appealed to them. The disadvantages may be enumerated, as follows: (1) The construction of supports to hold a track in a horizontal position, perhaps 6 or 8 feet above a slanting rock surface, requires considerable timber and many hours of labor, part of which is done by skilled men. (2) There is a great loss of time in starting and straightening channel cuts. The machine is so high above most of the rock surface that the channel bits, no matter how tightly they are clamped,

swing to one side or spring when they strike the rock, so that starting or maintaining a straight cut is difficult. In practice the helper generally takes a hand tool and hammer and starts the cut. After a little progress with the machine further delay may be necessary to straighten the cut. Altogether much time may be lost before the machine may be worked continuously. (3) If the rock has a steep dip a short length of track only may be set up, usually resulting in loss of efficiency. In general, operations on a small scale are less economical than those on a large scale. A short channel cut is more expensive per square foot than a long one under similar conditions. (4) On account of the rapidly increasing height of the track above the rock surface as the machine travels in the direction of dip, one length of steel can be used over a small part of the course only, and hence short cuts with frequent changes of steel are necessary.

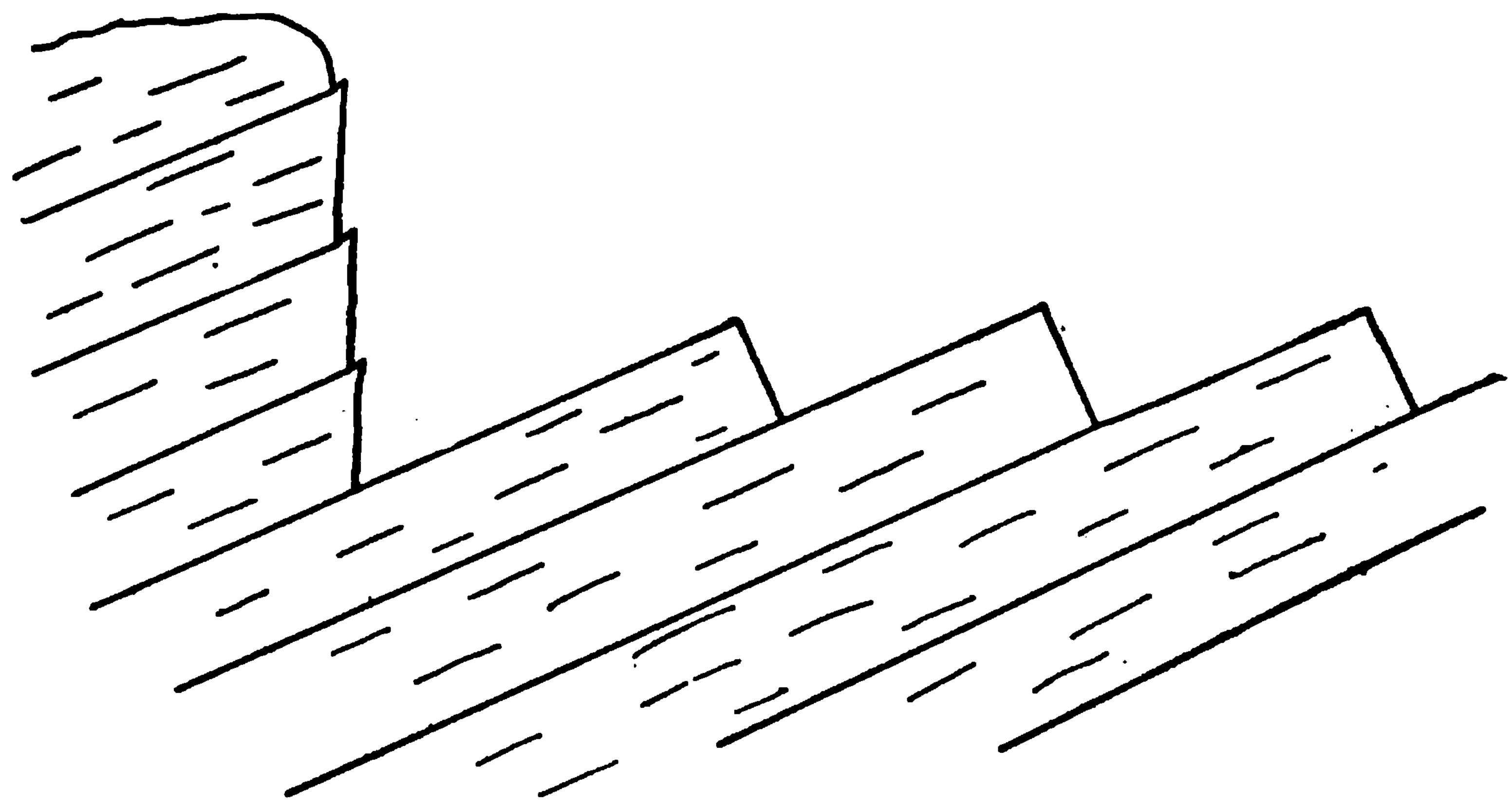

FIGURE 6.—A "saw-tooth" quarry floor.

(5) The necessity of cutting out numerous corners may increase the proportion of lost time.

INCLINED-TRACK CHANNELING.

An improved method of quarrying on an inclined floor is to place the channeling-machine track on the inclined rock surface in the direction of the dip. If the floor is of the saw-tooth type, the upper beds should be worked down successively until a large area of rock surface is obtained parallel with the open beds, as illustrated in figure 7. The force of gravity, which would cause the channeling machine to run too rapidly down the grade and probably entirely prevent its ascent is overcome by the use of a balance car or balance weight. The balance car may be placed on a parallel inclined track and loaded with enough stone or iron bars to exactly counterbalance the weight of the channeling machine, to which it is attached by means of a cable running over a sheave. The sheave is attached to

the upper end of the track. This method is employed with success in several Vermont quarries.

A second method is to place the balance-car track on a slanting unused part of the quarry. Though requiring a long cable attachment the plan works well, and is used successfully in Alabama. There is a considerable saving of time and expense in having a permanent balance-car track.

A third device, which must be arranged with special care in order to avoid the danger of accident, is a counterbalance weight attached to a cable that passes over a sheave in the tunnel ceiling.

For light grades a hill-climbing device observed on some machines gives satisfaction. It consists of a cable that passes over a sheave at the upper extremity of the track and winds on drums on the truck axles. The device prevents the machine from slipping, but does not equalize the load like the balance car.

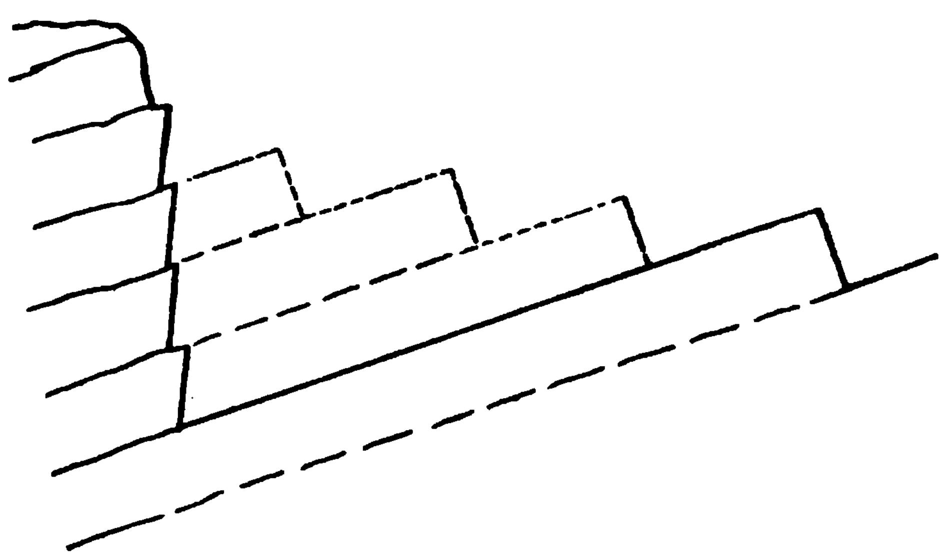

FIGURE 7.—Manner in which an inclined floor may be developed from a "saw-tooth" floor.

HIGH EFFICIENCY OF INCLINED-TRACK CHANNELING.

Quarrymen who have not had the experience may doubt the success of inclined channeling. However, many who have tried it have met with unqualified success. One Alabama company claims that channeling on a floor parallel with bedding that dips about 33° has resulted in a 50 per cent increase in efficiency over the saw-tooth method of operation.

In this connection the following quarry report of a company operating in Quebec, Canada, is of interest:

Monthly channeling report covering work of day gang in a Quebec quarry.

Machine No.	Marble excavated.	Time in operation.	Time moving.	Time under repair.	Time shut down.
	Square feet.	*Hours.*	*Hours.*	*Hours.*	*Hours.*
3	529.6	139.5	44.5	34	31
5	613.0	157	31	31	32
7	695.5	208.5	26.5	....	29
8	712.5	187	23.5	18.5	24
9	1,355.1	178	44	8	23

The good showing of machine 9 is the notable feature of this report. Machines 3, 5, 7, and 8 were operated on level elevated tracks on a saw-tooth floor. Machine 9 ran on an inclined track with a balance car. The average month's cutting per machine by the saw-tooth method was 637.6 square feet, whereas the one machine with the

balance car cut more than double that amount. Such figures offer convincing proof of the efficiency of inclined channeling.

With many quarrymen the physical difficulties of moving and operating machinery on inclined floors constitute sufficient cause for rejecting the method. However, it has been found that when men become accustomed to the new methods they work with the same facility as on level floors, and may even prefer the changed conditions. Methods of employing cables, snatch blocks, and hoists for handling heavy machinery are quickly devised by intelligent foremen. In one Vermont quarry inclined channeling is conducted successfully on a floor slanting 45°. The cost of channeling is approximately 14 per cent greater than on a level floor, and the cost of other operations shows a small increase.

When channeling machines are being operated on inclined tracks, no scaffolding is required. The tracks are placed flat on the rock surface with little more expense than when placed on a level floor. Much time is saved in the process of starting and straightening cuts, the field of operations need not be limited by short tracks, and the work is not interrupted by frequent changes of steel.

Plate V, *A*, shows a marble quarry with open beds dipping 24° to 28°. The present method is to channel in short cuts with the channeling machines working on tracks supported in level position. The inclined floor shown in the plate has an area of about 4,000 square feet which is sufficient to warrant the use of the inclined track and the balance car.

Even if conditions seem unfavorable the inclined-track method may prove to be the more economical. In one quarry observed the structure is a low anticline or arch, and the open bedding planes dip westward in the western part of the quarry, maintain an approximately horizontal position near the center, and dip eastward in the eastern part. The rock is now quarried on a level floor with the production of a large number of angular blocks. It is probable that the maintenance of a quarry floor parallel with the bedding, even though it involved varying angles of inclination in different parts of the quarry, would give better results.

INCLINED-TRACK CHANNELING ACROSS BEDS.

Figure 8, A, illustrates open marble beds dipping at an angle greater than 45°. In order that right-angled blocks may be obtained without a saw-tooth floor, the obvious method of channeling is to make the floor perpendicular to the beds as indicated in the figure. Such a method has not been observed in any American quarry, though it should be successful.

A. A QUARRY FLOOR SUITABLE FOR INCLINED-TRACK CHANNELING.

B. METHOD OF QUARRYING ACUTE-ANGLED BLOCKS.

C. CHANNELING DIAGONALLY ON A QUARRY FLOOR INCLINED 36°. A BALANCE CAR IS USED TO COUNTERACT THE EFFECT OF GRAVITY. QUARRY AT GANTTS QUARRY, ALA.

INFLUENCE OF RIFT OR COLOR BANDS ON CHANNELING.

Figure 8, B, illustrates marble beds dipping at an angle of less than 45° but with no open bedding planes. They are assumed to have, however, parallel with the bedding, a series of color bands or streaks of impurities or a decided rift, or the successive beds are of varying quality. The most desirable development of such beds is more complicated.

If, for artistic effect, it may seem desirable to have the color bands pass diagonally through the blocks, it may be advisable to excavate right-angled blocks on a level floor. Ordinarily, however, a proper classification of material would demand that the marble be split parallel with the bed, also if the material is uniform in quality, and possesses a decided rift, it can be worked more easily parallel with the bedding.

Figure 8, C, illustrates beds having rift, impurities, or color bands as represented in the beds shown in figure 8, B, but with the beds

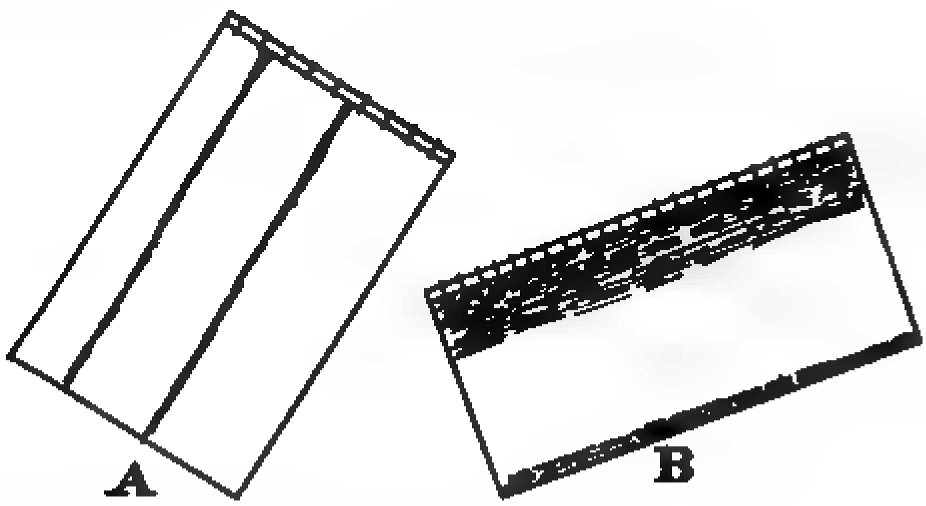

FIGURE 8.—Proper positions of channeling machine tracks for beds of different inclinations. A, open beds inclining at an angle greater than 45°; B, nonuniform beds inclining at an angle less than 45°; C, nonuniform beds inclining at an angle greater than 45°.

dipping at an angle greater than 45°. In developing such beds, if it seems advisable to use inclined channeling, the floor should be at right angles to the beds rather than parallel with them.

WASTE DUE TO ACUTE-ANGLED BLOCKS.

If steeply inclined marble beds are channeled on a level floor, and the cross breaks or cross channel cuts are made parallel with the bedding, acute-angled blocks will result, as is illustrated in Plate V, *B*. A complete discussion of the conditions under which such blocks may or may not be quarried economically is given in a later section dealing with the problem of waste. As pointed out in that section, if thin stock is desired, the waste resulting from acute-angled blocks may not be excessive. Saw gangs require longer blades however, and when longer blades are used the tendency toward unevenness in

the cuts is increased. If much cubic stock is desired undoubtedly it is more economical to quarry on a slanting floor parallel with the beds.

EXAMPLE OF INCLINED CHANNELING.

A large tunneled quarry operated in Vermont furnishes a good illustration of quarrying in conformity with bedding, and a detailed description of the various processes may be instructive.

Near the top of the quarry the beds dip about 70° east. At a depth of 50 or 60 feet, they curve back and dip west, and lower down again dip to the east, the angle of dip decreasing to about 30° near the bottom of the quarry. There are few open bedding planes. The excavation is about 300 feet deep, and long tunnels are driven in the direction of the strike.

Gray or variously colored bands occur at intervals, marking off distinct beds. The beds vary greatly in quality, some being almost pure white, and others gray, wavy, or banded. The various beds are named, and the characteristics and qualities are well known to the foremen.

In the upper part of the quarry, where the dip of the beds is steep, a level quarry floor is maintained. According to the rules given above, the quarry floor in this part of the quarry should be inclined to cut across the beds at right angles and thus avoid the production of acute-angled blocks. In this particular place, however, a level floor is preferable. As mentioned above, the beds near the top of the quarry dip to the east, at a lower level are vertical, and at still greater depth dip west. If the floor were to be maintained at right angles to the bedding the condition described would necessitate a constantly changing floor level, resulting in the production of many angular blocks.

Except for wall cuts, channels are made in one direction only—at right angles to the strike. The long masses of rock thus obtained are divided into small blocks by wedging in drill holes, projected in rows parallel with the bedding, as shown in Plate V, *B*. Effort is made to so place the holes that each block is of uniform quality.

Near the bottom of the quarry, where the dip of the beds is 45° or less, the quarry floor is maintained parallel with the bedding. A balance car on a parallel track is employed to overcome the effect of gravity, and thus permit the channeling machine to operate with the same motive power as on a level track.

The depth of the channel is governed by the position of those natural lines of separation that mark out a definite change in the quality of the rock. The channel cut may pass through one or two of these lines of division, but is always terminated by one of them.

If no open bedding planes appear, each of the bands of uniform material may, for convenience, be designated as a bed. The method of determining when the bottom of a bed is reached is of interest. The channeling machine, when running in its ordinary vertical manner, continually loses space at the end of its run, leaving a slanting end. Water is poured down this inclined end in order to wash it clean, and an incandescent light with a specially flattened bulb is let down. By this means, one who is thoroughly familiar with the rock may identify at a depth of 10 to 12 feet the bands that separate the various beds.

When drilling or channeling is done at right angles to the quarry floor, right-angled blocks are obtained. By working thus on an inclined floor, blocks of economical form are produced, and an exact classification of the product is made possible.

INFLUENCE OF DIP AND STRIKE ON DIRECTION OF CHANNEL CUTS.

The influence of bedding on the attitude of the quarry floor has been discussed in detail. The influence of dip and strike on the direction of the channel cuts is also of great importance. If vertical channel cuts make oblique angles with the strike of a marble deposit in which separation of blocks must be made along the planes of steeply dipping beds, it is obvious that blocks having too much of an acute angle will result. It is generally more convenient and more economical to make channel cuts parallel with the strike. Exception may be made if pronounced joint systems meet the strike of the rock at oblique angles. Under such circumstances channeling parallel with the joints rather than with the strike is justifiable. However, an inclined floor should be maintained, and the cuts should be at right angles to the floor in order that rectangular blocks may be produced.

CHANNELING IN RELATION TO UNSOUNDNESS.

CHANNELING PARALLEL WITH JOINT SYSTEMS.

In a previous section (pp. 22–29) a discussion is given of the causes of unsoundness and of the various forms in which it appears. The most important feature of joints in relation to channeling is their occurrence in more or less definite systems. The importance of recognizing such systems and quarrying in accordance with them can scarcely be overestimated. In quarries in which joints are prominent the quarrymen should endeavor to make their channel cuts parallel with the chief joint systems. Blocks that are intersected by oblique joints are almost useless. If, on the other hand, the joints parallel one pair of faces, the waste is greatly reduced.

CHANNELING COINCIDENT WITH JOINTS.

Paralleling the joint system is only one step in economical channeling. If it be assumed that the joints intersecting the blocks are parallel with one pair of faces, considerable waste may still result. If saw cuts are made parallel with the joints, two or three slabs only may be wasted. Usually, however, joints cut across the grain and slabs are cut parallel with the grain. A single joint, therefore, will intersect every slab. Consequently, economy demands that channel cuts not only run parallel to joints, but that they be spaced in such a manner as to reduce to a minimum the number of joints contained within the blocks. In other words, whenever possible the channel cuts should coincide with joints. If joints are spaced at irregular intervals, channel cuts should be spaced in the same way. It is unreasonable to maintain that the production of blocks varying in size is undesirable. The presence of irregularly spaced joints will not permit the quarrying of uniform sound blocks, and it is clear that in the shaping of any structural design sound blocks of various sizes will cut to better advantage than unsound blocks of uniform size.

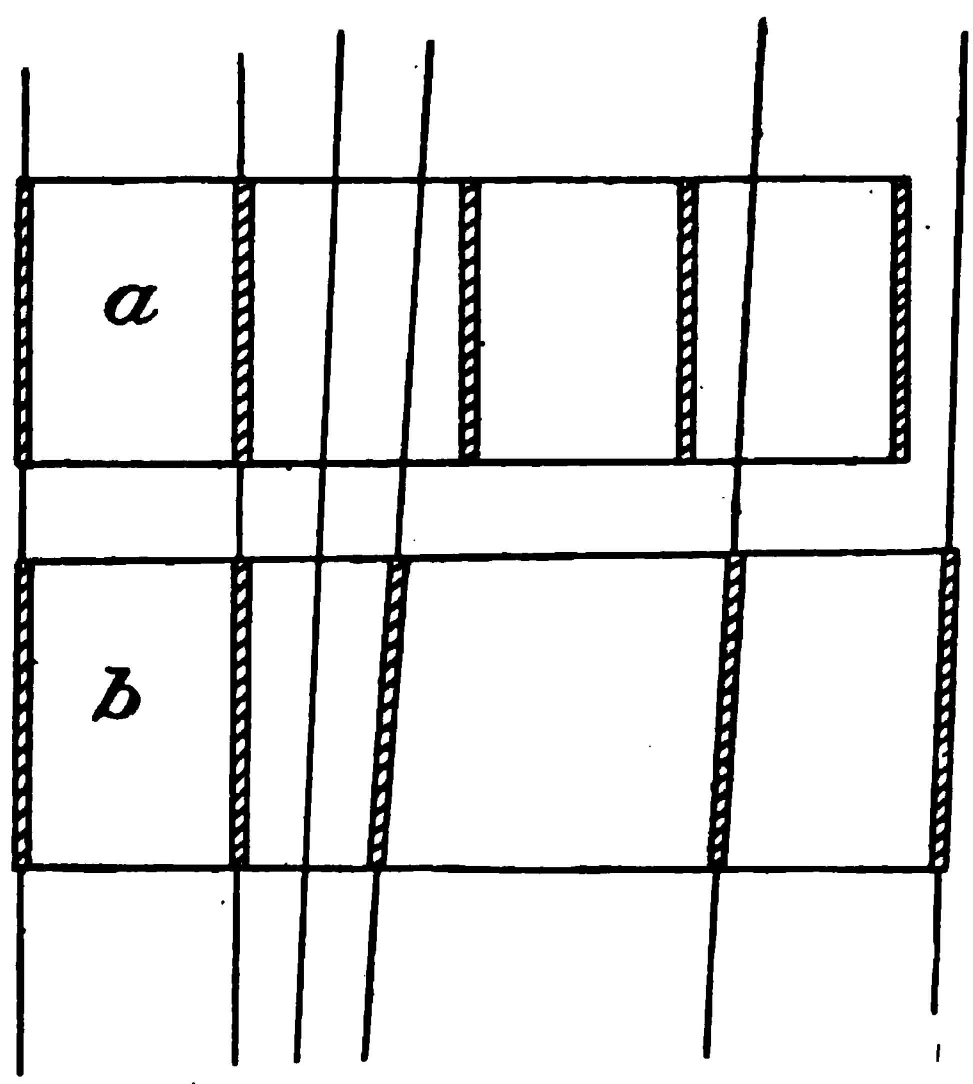

FIGURE 9.—The right and the wrong way of channeling unsound marble: *a*, Channel cuts equally spaced, involving waste; *b*, channel cuts made coincident with joints, avoiding waste.

The advantage of eliminating joints by channeling coincident with them is illustrated in figure 9. Six unequally spaced joints are shown in an approximately parallel system. In the figure *a* represents equal spacing of channel cuts without regard to unsoundness. Of the four blocks that result, one is intersected by one joint and another by two joints, and two blocks are sound. At *b* is shown spacing varied in such a manner as to make the channel cuts fall on the joints. As a consequence one small block is intersected by a joint, and three blocks, one of which is exceptionally large, are sound. A judicious spacing of channel cuts may thus tend to produce a large proportion of sound stock.

ECONOMY OF CHANNELING IN ACCORDANCE WITH UNSOUNDNESS.

Figure 10 shows the economy of channeling in accordance with unsoundness. Two right-angled systems of unequally spaced joints are shown. They are identical in both parts of the figure. As shown

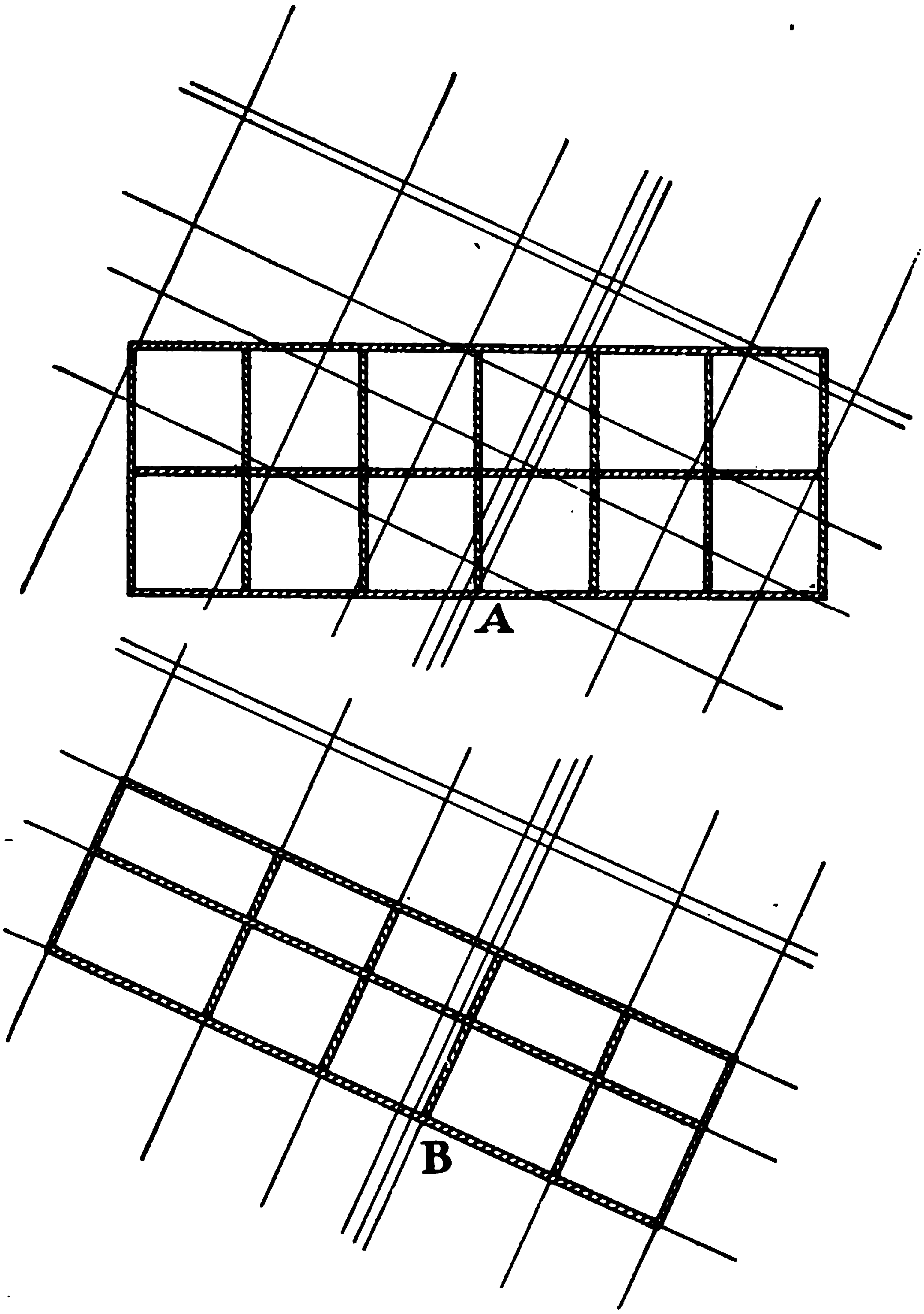

FIGURE 10.—The economy of channeling in accordance with joint systems: A, Plan of channeling without reference to direction or spacing of the joints, involving great waste of marble blocks; B, plan of channeling in which channel cuts are made parallel with, and as far as possible coincident with, joints, avoiding waste of blocks.

in figure 10, A, the channel cuts make oblique angles with the joint systems. The channel cuts are spaced regularly in both directions without regard to unsoundness. As a consequence, every block quarried is intersected by one or more joints.

Figure 10, B, illustrates the remarkable improvement that results from changing the direction of channel cuts to parallel the joints, and from varying the spacing of the cuts so that they fall exactly on the joints. Of the 10 blocks that will be produced 8 will be sound and 2 will have two parallel joints close to one side. The advantage of cutting out blocks in accordance with unsoundness is so apparent that it is remarkable how few quarrymen have made practical application of the principle.

UTILIZATION OF JOINTS FOR MAKING CROSS BREAKS.

Channeling may be done in one direction only, the cross breaks being made by some other means, or the rock may be channeled in two directions. In many marble deposits, one system of joints is pronounced, and cross joints are few in number. Under such conditions, it is wise to channel in one direction only, at right angles to the chief joint system. Advantage may thus be taken of the joints in making cross breaks. A long mass of marble can be channeled at the back and ends, and wedged up at the floor until it is free. The wedges near one end may then be driven hard and the others left untouched, producing a bending strain which may be sufficient to cause the mass to break at one of the joints, which is a plane of weakness. Heavy wedging at both ends, the middle wedges being left untouched, may be even more effective.

AN ILLUSTRATION OF CHANNELING IN ACCORDANCE WITH UNSOUNDNESS.

An excellent illustration of a modification of channeling directions to parallel strike, dip, and unsoundness is to be found in a certain quarry in Vermont. One channel cut parallels the strike, which is N. 5° W. The beds dip about 70° to the east, and, on account of a green banding parallel with the bedding, the proper classification of the material and the production of attractive patterns in the banding demands that the channel cuts be inclined to parallel the beds. A remarkable system of parallel open joints (see fig. 2, p. 26) intersects a limited part of the quarry. The joints run N. 82° E. and dip 61° N. The second channel cut is made parallel with these joints. Thus the channel cuts meet at an angle of 87°, and are both inclined from the vertical. The maintenance of a level quarry floor is justified by the fact that such conditions prevail over part of the quarry only. By thus quarrying in accordance with rock structures, sound blocks with a desirable color distribution are obtained.

A short distance north of this quarry is another, which is now about 50 feet deep. The system of joints referred to above does not appear as yet in the excavation. As the joints dip 61° N., it is an easy matter to determine the point at which, with continued excavation, they

may be expected to appear. The present plan of quarrying is to channel vertically at right angles to the strike, and to make the cross breaks by wedging in drill holes parallel with the bedding. When the zone of parallel joints is reached, a modification of the method will be justified.

A PROPOSED IMPROVEMENT IN QUARRY METHODS.

As another illustration of quarrying in conformity with unsoundness, reference may be made to a certain marble quarry in which a change of plan is contemplated. The prevailing joints run north and south, and a second series of joints filled with crystalline calcite, known locally as "glass seams," run east and west.

The walls of the present excavation run approximately northeast and northwest. Thus a majority of the joints intersect blocks diagonally. When the present excavation is extended it is proposed to make the cuts in north-south and east-west directions, and thus parallel the chief rock structures.

INDISTINCT JOINTS AND JOINT SYSTEMS.

In the preceding paragraphs reference has been made only to those joints or joint systems that are easily recognizable. In quarries in which joint systems are obscure, the problem of economic channeling is more difficult.

The difficulty of recognizing joint systems is due to two main causes. The joints may be rather easily seen, but so irregular and intersecting at so many angles that system seems to be absent, or they may be so indistinct that only skilled men with long practice can recognize them in the quarry. When joints are both indistinct and irregular the problem is complex.

HOW TO DEAL WITH COMPLEX JOINTING.

A desirable method of approaching such a complex problem can be best appreciated by consideration of a concrete example. The operations of an Alabama marble company offer one of the best illustrations obtainable of the means that may be employed to discover systems of unsoundness and of how quarry methods may be modified later in accordance with these systems.

The quarry operations are greatly hampered by the presence of cracks that are locally termed "headers," many of which are almost unrecognizable in the quarry. However, when the blocks are sawn they cause the resulting thin slabs to break into small and angular pieces, many of which can not be used. So serious was this difficulty that profitable operation seemed impossible.

The marble was first quarried on a saw-tooth floor parallel with and at right angles to the strike. The beds dip about 33°. Blocks were intersected by joints and also by mica bands which run parallel to the bedding, and the waste was excessive. The proportion of sound stock was increased to some extent by quarrying on an inclined floor parallel with the bedding. Even with this improvement the presence of many "headers" intersecting the blocks obliquely resulted in great waste.

In an attempt to further overcome the difficulty the quarry floor was laid out in sections, and the marble blocks as they were quarried were all so numbered and oriented that the section to which they belonged and their exact position in the section could be determined. The visible "headers" were located accurately, and diagrams were made showing their position. The "headers" that could not be located in the quarry were found in the slabs after the blocks had been sawed. As every block had been oriented and as the original position of the slab was known, reference of the slab to its original position in the quarry indicated the point at which the joint must cut the quarry wall. As it was known that the joints must be present at certain points, diligent search was made to find evidences of them on the quarry walls. Only persons skilled in such observations through long practice can recognize obscure joints. In the quarry under discussion one efficient quarryman is detailed for the express purpose of finding and marking them on the quarry walls and floors.

The information thus obtained was supplemented by study of a drill core obtained by projecting a hole down the dip with a double core-barrel drill. A diagram of all the "headers" thus discovered on one of the most unsound floors is shown in figure 11. As may be seen from this figure, the joints in general follow two distinct systems. These two sets of joints have for convenience been designated *x* cracks and *y* cracks. The general direction of the *x* cracks is S. 80° E. They are distinctly visible, regular, and sufficiently far apart to allow the production of fair-sized blocks. They are all normal faults, having a slip of less than 1 inch. The *y* cracks are those mentioned above as being very indistinct. Their direction is N. 24° E. They are closely spaced, branching, and fade out and reappear irregularly. They are not faulted, or at least there is no apparent slip. They occur in zones about 50 feet wide, and these are followed by zones 60 feet or more in width in which the *y* cracks are absent or few in number.

In addition to the *x* and *y* cracks there is a series termed "slick seams," which run in the direction of the dip. They are probably caused by expansion and contraction from solar heat. They disappear at a depth of about 75 feet. A few joints are noted in the direc-

tion N. 82° E. The strike of the rock is N. 46° E., and the dip averages about 33° SE.

As indicated in figure 11, the two main systems—the *x* and the *y* cracks—are 76° apart, and the *x* cracks make an angle of about 54° with the strike.

With this arrangement of headers, four plans of channeling were possible, as follows:

(1) Channels parallel with and at right angles to the strike.

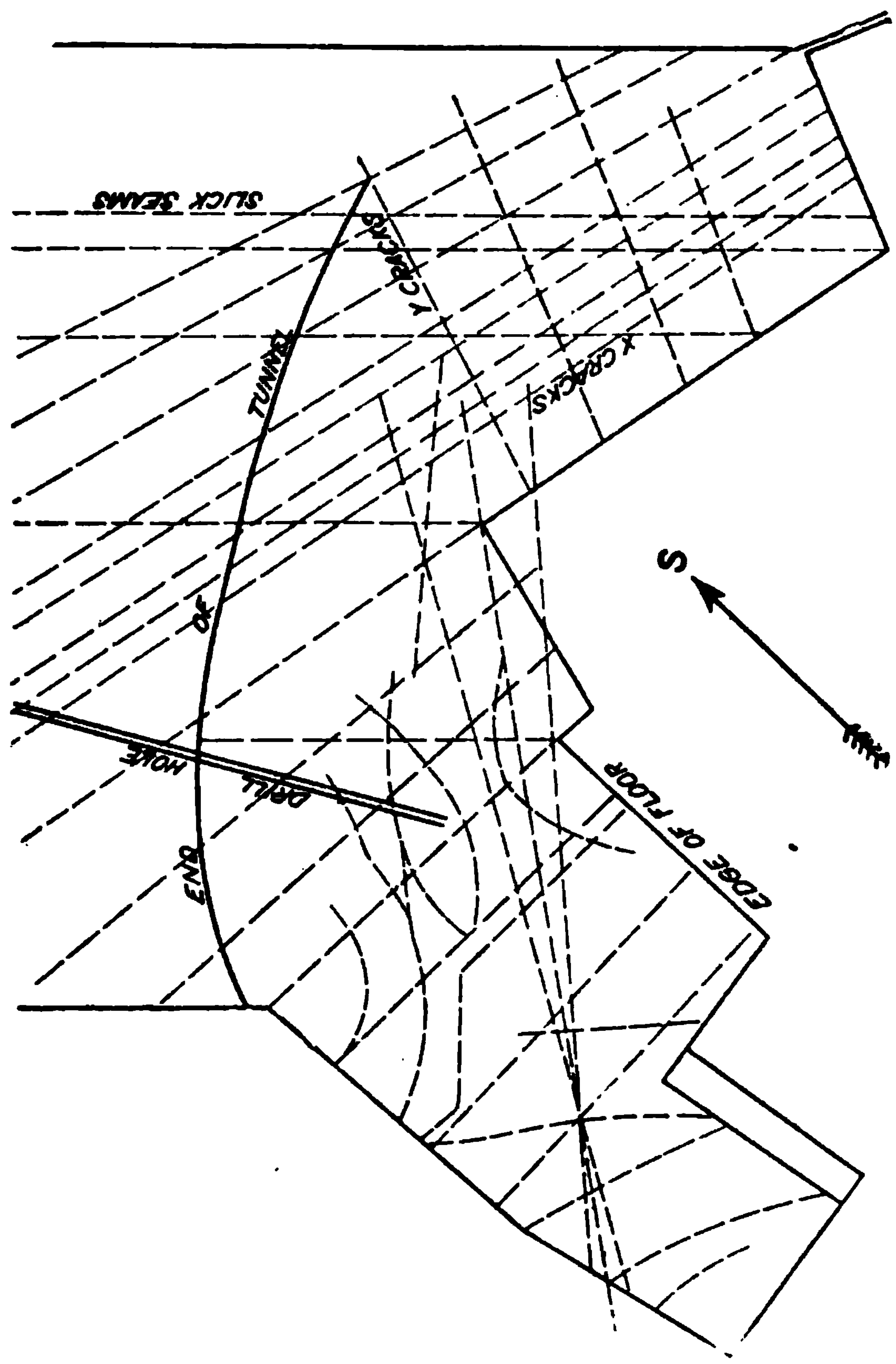

FIGURE 11.—Diagram of joint systems in one bed of an Alabama marble quarry.

(2) Channels parallel with and at right angles to the *y* cracks.

(3) Channels parallel with and at right angles to the *x* cracks.

(4) Channels parallel to the *x* and *y* cracks.

Worked out mathematically, plan 4 is the best, but only slightly better than plan 3. Where quarrying is now carried on, the *y* cracks are not abundant, and plan 3 is now followed. If the *y* cracks become more numerous with further development, plan 4 may be adopted.

The conditions prevailing on this floor were exceptionally bad. With the most economical methods only 15 per cent of the marble quarried was in sound blocks. On the next floor below 30 per cent was sound, and on the third floor 35 per cent. Quarrying parallel with and at right angles to the strike on the floor mapped would give no sound blocks, and probably few would result from a similar method on the lower floors.

It is established beyond question, therefore, that the most economical method of quarrying in this deposit is to channel diagonally on a slanting floor. To quarrymen who would hesitate at quarrying on an inclined floor under favorable conditions such a method must seem impractical. However, its success in this particular quarry is assured. Although the difficulty of operation is undoubtedly great, the advantages more than offset the increased cost per square foot of channeling. Plate V, *C*, illustrates a channeling machine operating on a track placed diagonally on an inclined floor.

The conditions in the quarry mentioned above are about the most complicated that can occur, as the joints are not only seemingly irregular, but many of them are indistinct. If the most complicated conditions can thus be handled satisfactorily simpler problems should be comparatively easy of solution.

CUTTING WITH UNSOUNDNESS IN TENNESSEE QUARRIES.

In some of the Tennessee quarries there are distinct joints known as "cutters," and other cracks which are indistinct, appearing as fine white lines. The latter are known as hair lines. Though the hair lines are difficult to recognize in blocks or in the quarry, thin slabs break readily along these planes.

In some quarries the distinct cutters show a prevailing system and the hair lines a second system nearly at right angles to the former. Under such conditions the rock may be quarried to advantage if the cracks are not too close together.

An excellent example of quarrying in accordance with rock structure and unsoundness is to be found in a quarry near Knoxville, Tenn. Open bedding planes dip 15° to 20° N., "cutters" occur in a predominating east-and-west system, and indistinct hair lines run generally north and south. The rock is channeled north and south, drilled east and west, and quarried on a slanting floor, thus paralleling the three prominent rock structures.

POSSIBLE OBJECTIONS DISCUSSED.

Some quarrymen would condemn a new method immediately if it were found that a decrease in the rate of channeling resulted. Such action would indicate faulty reasoning. Quarrymen must realize

fully that their purpose is not to produce channel cuts and drill holes, but blocks of sound marble. If by a new method a greatly increased proportion of sound stock is obtained at a nominal increase in cost of production, the method is successful even though the rate of channeling is greatly reduced.

To properly search out and map unsoundness requires considerable time and energy. However, the far-seeing quarryman realizes that such labor, although temporarily unproductive, prepares the way for more economical and efficient operations. It is important that quarrymen should recognize the undoubted desirability of such preliminary investigation and the modification of methods to suit the conditions.

METHODS OF SEARCHING FOR INDISTINCT JOINTS.

If joints are very indistinct, a method of search devised by a Tennessee marble company is noteworthy. As all quarrymen know, only distinct joints can be recognized on a channeled surface. They are much more easily recognized on a fractured surface. Hence, before making cross breaks on a long channeled mass the quarryman takes a hand tool and a hammer and "points" a strip about 6 inches wide in a horizontal direction across the channeled surface. The effect is to produce a band having a fractured surface on which the natural joints are easily detected. The cross breaks are then made where the joints appear, and much of the unsoundness is in consequence eliminated from the finished blocks.

The same method is followed on the quarry floor. "Pointed" bands are made in two directions at right angles and 10 to 12 feet apart. The joints are thus located and the channel cuts can then be made to best advantage.

In a certain Alabama quarry small pieces are broken off the corner along the working edge of a quarry floor, thus exposing fractured surfaces. When water is thrown on the surfaces the indistinct joints appear as fine white lines. Channel cuts are then spaced to coincide with the joint lines wherever possible.

THE BOWLDER QUARRIES OF TENNESSEE.

Unsoundness in certain Tennessee marble quarries is greatly emphasized by erosion. Running water has entered the minute fractures and so worn them that large cavities have been formed between which the rock stands up as pinnacles and spires. The cavities are now filled with clay. Such a condition constituted what is known in Tennessee as a bowlder quarry. As the erosion began in planes of weakness it is probable that the masses of rock that now remain are comparatively sound. In such quarries little if any channeling is necessary.

STUDY OF UNSOUNDNESS IMPORTANT.

The suggestion in the section devoted to prospecting and developing marble deposits that a careful preliminary study of unsoundness should be made before operations are commenced on any prospect will be more fully appreciated when quarrymen have studied the preceding paragraphs and noted the vast influence unsoundness exerts on the proportion of waste produced. It is probable that greater losses result from quarrying without regard for unsoundness than from any other single cause. Present operators may in many instances greatly increase the proportion of sound stock quarried by making careful study and detailed diagrams of all visible unsoundness and then seeking methods of excavation that are best in accord with the conditions. Sufficient examples have been cited to convince the most doubting of the economy of such methods.

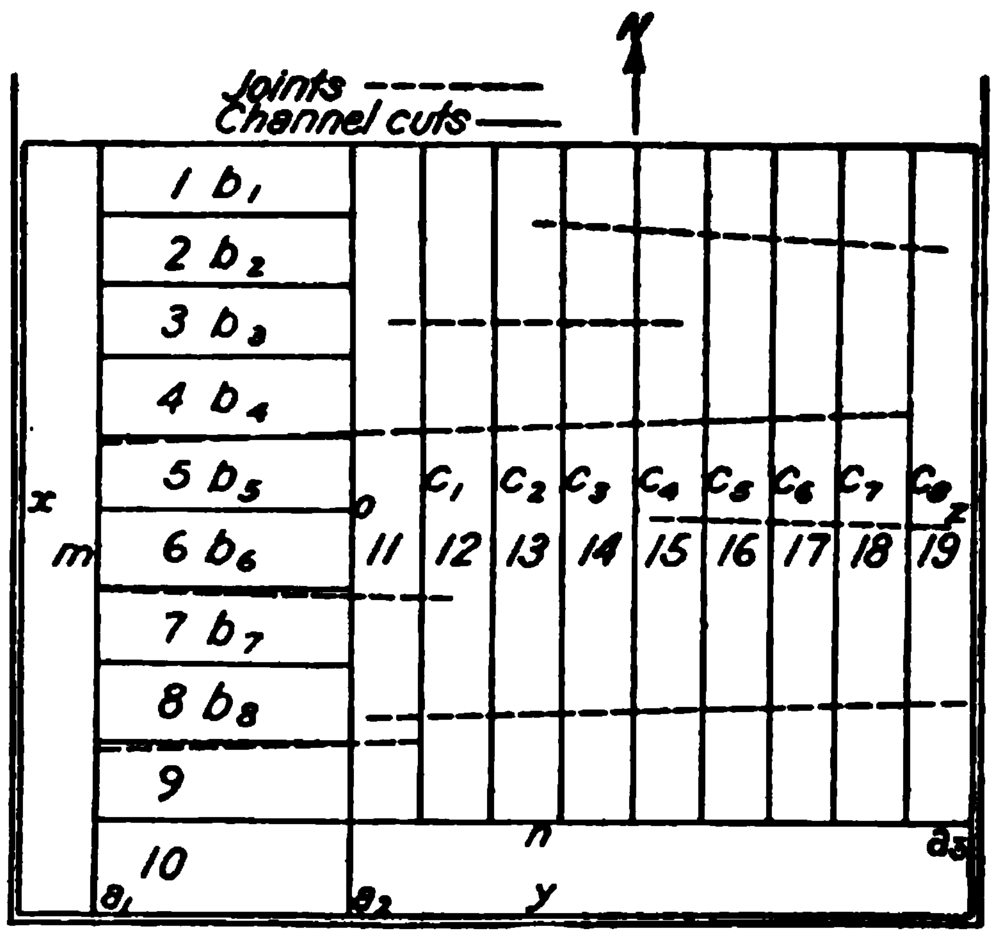

FIGURE 12.—Method of channeling a quarry floor employed by a Georgia marble company.

THE QUARRY OPERATOR'S PRIME OBJECT.

In concluding the discussion of channeling in accordance with bedding and unsoundness the author desires to emphasize the fact that the prime object in marble quarrying is not to establish high records in rate of channeling, or in gross production per man per month, irrespective of the form or quality of the product, but rather to produce rectangular blocks of sound marble. This object should be kept prominently in view, and should govern the choice of machinery and the method of rock removal in every instance.

REDUCTION IN NUMBER OF CORNER CUTS.

All operators of channeling machines are familiar with the loss of time resulting from the necessity of cutting out corners when wall cuts are met by subsequent channel cuts. Methods of channeling should be devised that tend to reduce to a minimum the number of corner cuts required. The plan followed by a Georgia marble company may well be cited. Figure 12 shows the position of channel cuts on a bench that is open to the north. The first step is to make the

wall cuts, x, y, and z. Next, the two cuts, m and n, are made. Their purpose is to remove the necessity of cutting out numerous corners. By making these cuts only three corners, a_1, a_2, and a_3, must be cut out. After the cut o has been made, the short cuts b_1 to b_8 follow, and finally the cuts c_1 to c_8. The key blocks *1* to *10* are broken free successively by gadding and wedging, beginning at the open edge of the bench. When these blocks have been removed, blocks 11 to 19 are successively raised and removed. It will be noted that the prevailing joints run approximately east and west. Blocks 11 to 19, when wedged up from beneath, break along joints. This plan has the double advantage of making cross drill holes unnecessary and at the same time of reducing to a minimum the number of joints that intersect blocks. Blocks of considerable length free of joints may be used for columns.

CHANNELING IN RELATION TO RIFT.

In certain marble deposits there is a tendency for the rock to split with greater ease in one particular vertical plane. Ordinarily, under such conditions, channel cuts should be made at right angles to the direction of the rift in order that advantage may be taken of this ease of splitting, thus facilitating the making of cross breaks. If no rift exists, the difficulty of obtaining a straight and even break may be so great as to justify channeling in both directions.

USE OF WIRE SAW IN QUARRIES.

MECHANISM OF THE WIRE SAW.

The wire saw consists of an endless wire rope about one-fourth inch in diameter and composed of three strands. The wire passes around a driving wheel and is carried on pulleys to that part of the quarry where cutting is to be done. A sliding carriage is placed at a convenient point in the circuit to give the necessary tension to the wire. Renwick[a] states that in France and Belgium holes 2 to 3 feet in diameter are cut by means of cylindrical core drills. The cores are removed and standards erected in the holes. Pulleys over which the wire runs are attached to these standards.

THE PENETRATING PULLEY.

In early days much difficulty was experienced in guiding the wire in a slanting direction, but this difficulty was overcome in 1898 by the invention of the "penetrating pulley." This device is thus described by Renwick:[b]

> The pulley consists of three parts—(1) the pulley: This is a steel disk 19.6 inches in diameter and 0.27 to 0.31 inch thick, grooved on its edge to receive the wire. On

[a] Renwick, W. G., Marble and marble working, London, 1909, p. 34.
[b] Renwick, W. G., Op. cit., p. 38.

the central part of the disk and projecting on each side is a boss supporting a steel axle 0.97 inch long. (2) The fork: This takes the shape of a hollow steel bar 2.4 inches in diameter, grooved at the lower end to receive the pulley, and which can be lengthened by a series of tubes of similar diameter. The bottom of the fork acts as a bearing, on which the pulley runs. (3) The carriage: This consists of a standard which can be placed in position by three screws or guys. The standard carries a screw and drum attachment, by which an automatic progressive motion is given to the fork from the rotation of an exterior pulley, on which the helicoidal wire runs.

The operation of extracting stone is as follows: Two holes are first sunk to receive the fork on which the pulley runs. The standards carrying the wire having been placed in position and the wire set in motion, the penetrating pulley is brought close to the rock. The thickness of the pulley being slightly less than the diameter of the wire, the latter projecting from the edge of the pulley wears into and bites the rock, forming a groove into which the pulley enters. Sand and water are fed to the descending wire, and, the fork supporting the pulley following the hole made to receive it, the wire is carried through the rock from one standard to the other, making a cut in the direction required. Crosscuts are obtained by altering the position of the standards. Cuts can be made along a length of 10.9 to 16.3 yards, and for an equal depth, the progress of the cut being 1.56 to 5 inches per hour, this depending on the nature of the material and the length of the cut.

INSTANCE OF USE OF WIRE SAW IN QUARRY.

Although the wire saw is used extensively by marble-finishing plants, its use in American quarries has, during recent years, at least, been confined to certain Colorado quarries. In these quarries it has been found that the most useful application of the wire saw in quarry work is in the cutting out of masses of rock situated between two shafts. In the early days of these quarries three shafts 60 to 80 feet apart were opened in the mountain side. After they had been carried down to a considerable depth it was found advantageous to remove the intervening masses and to develop a single large opening. By means of a core drill a hole was projected from one excavation to the next at a point close to the ceiling. The wire was passed through this hole, fitted around the necessary pulleys, and spliced to form a continuous belt. Steel shot and water were used as abrasive.

A mass 4 feet wide and 5 feet high and extending from one opening to the other was first removed to give a working space. Both wall and ceiling cuts were made with the wire saw. The ceiling cuts, were somewhat slower and more difficult to make than the wall cuts, as the abrasive would not feed readily to the saw. A second horizontal cut was made about 18 inches below the ceiling cut. The intervening mass was broken up and removed as waste in order to gain sufficient room for the removal of blocks without jamming. On account of the narrowness of the saw cuts, great difficulty is often encountered in removing this material. It may be noted, however, that a great saving of marble is thus effected, as in ordinary tunneling the preliminary opening is 6 or 7 feet high. Additional vertical cuts were

spaced in the same manner as those made with a channeling machine, whereas the cross and the floor breaks were made by drilling and wedging.

It has been found that the cutting is just as effective if the abrasive is fed to the saw at one point only, the point where it enters the rock, as when fed at several points along its course through the rock.

DESIRABILITY OF USING LONG WIRE.

It is important that a long wire be employed. The wire is abraded during the process of sawing and continually becomes smaller until at some point it becomes too weak to withstand the strain and breaks in consequence. It is obvious that by increasing the length of the wire the rate of wear on any given part is correspondingly diminished. One company uses a wire nearly a mile long. Formerly the superfluous wire was passed around a drum in numerous turns. It has been found more satisfactory, however, to run the wire in a simple turn over a pulley situated at a distance from the quarry.

METHOD OF REPLACING AN OLD WIRE BY A NEW ONE.

In the early days of wire-saw operation much time was lost when a wire was worn out before a cut had been completed, as the opening cut by a wire gradually becomes narrower as the wire is reduced in size by abrasion. When the wire was replaced by a new one much of the cutting had to be repeated, as the opening was too narrow for the new wire.

Recently a method has been employed that obviates this difficulty. When the wire has been worn small and appears nearly ready to be discarded, the force that crowds it against the bottom of the cut is relaxed, and it is allowed to run freely for some time. It thus wears an opening larger than its diameter. The new wire can be drawn through this opening, and the cut continued with little loss of time.

DISADVANTAGES IN USE OF WIRE SAW.

When working normally the wire saw cuts much more rapidly than a channeling machine and requires less attention.

The presence of flint balls in marble is, however, a serious obstacle to the successful operation of the wire saw. On account of its extreme hardness the flint will greatly diminish the speed of cutting or may entirely suspend progress. Moreover, the wire is likely to pass around the mass of flint, and in doing so, may be offset 3 or 4 inches from its regular course, resulting in an uneven surface.

Another disadvantage in the use of the wire saw is the obstruction of other operations by the wires that pass through the quarry.

They interfere more particularly with the pulling and the hoisting of blocks. Seemingly, the wire saw is best adapted for cutting out masses of rock that intervene between two shafts. Its advantage over the channeling machine in the ordinary process of quarrying has not yet been demonstrated in any American quarry.

DRILLING.

MACHINERY.

Ordinarily in quarry drilling vertical holes are made with a tripod drill or bar, holes in the face in horizontal rows with a quarry bar, and in vertical or inclined rows with a gadder. A bar is better than a tripod drill for projecting rows of holes in a straight line, as it saves much time in moving.

Recently the jackhammer has replaced both bars and tripod drills in several quarries. It has a mechanical rotating device and may be run as a one-man drill. It uses hollow steel. It is run dry, and part of the exhaust passes down inside the drill and blows out the rock dust. On this account it is best operated by air, as steam condenses and forms a mud which is removed with some difficulty. When operated with steam it ceases to be a one-man drill, as a helper is required to pour water down the hole.

The jackhammer is operated as a hand drill, and is a great timesaver, as a few seconds only are required to change steel or to move to a new hole.

In one Alabama quarry jackhammers have now replaced all bar and tripod drills. In 1913 and 1914 tripod and bar drills were used, and in 1915 jackhammers took their places. The increased efficiency that results is shown in the following table:

Drilling record of a quarry in which jackhammers superseded tripod and bar drills.

Period.	Linear feet of drill hole cut per month.	Cost per linear foot.	Cutting rate in 10-hour day.	Cutting rate in 10 hours of actual cutting.	Time efficiency.		
					Proportion of time drilling.	Proportion of time lost in repairing.	Proportion of time lost in moving.
			Linear feet.	*Linear feet.*	*Per cent.*	*Per cent.*	*Per cent.*
1913	[a] 11,981	$0.0780	99.7	132.6	78.63	0.70	20.43
1914	[a] 8,699	.0682	106.0	135.0	79.00	1.09	19.91
1915.							
January	6,022	.0404	90.0	106.2	88.74	0.91	10.35
February	7,203	.0332	105.4	111.0	95.02	0.58	4.39
March	7,760	0392	113.0	117.0	96.50	0.43	3.06

[a] Average of 12-month period.

The cost per linear foot in March, 1915, was just half as much as the average for 1913, and 40 per cent less than the average for 1914. This reduction in cost was due partly to the employment of one

instead of two men for each drill, and partly to the increased time efficiency indicated. In 1913 and 1914 the drills were operating on an average only 79 per cent of the time, whereas about 20 per cent was required for moving. In March, 1915, the jackhammers were actually at work 96.5 per cent of the time, and only 3 per cent was required for moving.

Other hollow-steel drills of similar type are now on the market, but are not in as general use as the jackhammer, and no figures concerning their efficiency were obtained.

A highly efficient bar drill employed by a Georgia marble company is used for both vertical and horizontal rows of holes in the quarry face. The bar is more than 12 feet long, and two drills are attached to it. In making a horizontal line of holes, each driller completes 6 feet of holes, and then the whole outfit is moved 12 feet, and the process is repeated. In drilling the face, two vertical rows of holes are projected simultaneously, one drill operating on each row. At the completion of each pair of holes the bar is moved up to the proper position for the next pair. Much time is thus saved as it takes no longer to adjust the bar for two drills than for one.

With the exception of the jackhammer, most drills work successfully with either steam or compressed air. If electric power is available, drills of special design may each be operated by a portable motor-driven pulsator. This method is especially advantageous if drilling operations are carried on at a considerable distance from the power plant, as power can be transmitted with greater ease by wire than by pipe lines or hose, and with less loss of power through leakage, friction, or heat radiation.

DRILLING IN RELATION TO CHANNELING.

The separation of rock masses is ordinarily less expensive per square foot by drilling and wedging than by channeling. The former method can be employed, however, only where a channel cut allows freedom of motion of the block in the direction in which the wedges tend to force it. Thus a certain amount of channeling is unavoidable. If there is a decided rift in the marble, drill holes may be placed much farther apart when splitting is being carried in the direction of the rift than when the splitting is carried across the rift. On this account when the wall cuts are made, if the direction of channeling is not influenced by bedding or unsoundness, one set of channel cuts should be made in a direction at right angles to the rift, and the cross breaks made by drilling and wedging. Advantage is thus taken of the rift direction to reduce the number of drill holes. As regards "liver rock," a term applied to a marble that has no rift, the necessity for placing drill holes close together and the difficulty of obtaining

a uniform break if the plug-and-feather method is employed, may render it advisable to channel in both vertical directions and to wedge only on the floor.

ARRANGEMENT AND SPACING OF HOLES FOR CROSS BREAKS.

If the rock is uniform and sound drill holes may be spaced regularly in vertical holes. The spacing of drill holes varies from 4 inches to 2 feet, depending on the ease of splitting the marble. When the marble is unsound or lacks uniformity in color or texture, it may be necessary to make slanting cross breaks in order to avoid waste and to properly grade the product. Such breaks are shown in Plate V, *B*, p. 58.

DRILL HOLES FOR FLOOR BREAKS.

When channeling is done in one direction only, except for wall cuts, the long masses of marble are broken loose from the floor before cross breaks are made. Occasionally a floor cut is made with a channeling machine, but almost invariably they are made by drilling and wedging. As with holes for cross breaks, the spacing of the holes is governed by the ease with which the marble splits. A common practice is to make alternate holes shallow and the intervening holes the full depth of the break desired. The depth of each hole is marked on the rock surface for the guidance of the worker in selecting wedges. The holes may be parallel with each other or may radiate in fanlike form.

PROPER SIZE FOR DRILL HOLES.

Drill holes should be as small size as possible without detracting from wedging efficiency. Most drill holes are made 1½ to 1¾ inches in diameter. It has lately been demonstrated that the diameter of drill holes can be greatly reduced without interfering with the successful operation of wedges. J. P. McCluskey, quarry superintendent of the Alabama Marble Co., found that on an average 4½ minutes was required to drill a 1⅜-inch hole 2 feet deep. The time required to drill a 1⅛-inch hole 2 feet deep was only 2½ minutes. All drill holes in the quarry of the company mentioned are now made 1⅛ inches in diameter. The change from 1⅜ to 1⅛ inches has resulted in a reduction of 44 per cent in the time consumed per linear foot of drilling.

In order to wedge successfully in holes of such small diameter, wedges of special design are used. They are described subsequently under the title "Wedging."

ADVANTAGE OF REAMING DRILL HOLES.

Acting on a suggestion made by the writer, a certain marble company tried the experiment of using a reamer to cut grooves on the opposite sides of drill holes, thereby assisting the splitting so materially that

fewer drill holes were required. The reamer was made in the form of a drill with wings at the sides sufficient to cut grooves about one-fourth of an inch deep on opposite sides of the drill hole. It could be used only in a reciprocating drill. When the hole was completed the drill bar was removed and the reamer put in its place. The rotating device was thrown out of gear, and by means of a bar through which the square shank of the reamer passed, the latter was held in proper position to cut the grooves exactly in line with the desired direction of splitting.

It was found that a straight break could be made with drill holes at least twice as far apart as when no reamer was employed. However, no reamers could be found that would bear the work required of them. The projections invariably broke off after short service. Until this mechanical difficulty has been overcome, the method is not considered to be practical.

SHARPENING AND TEMPERING OF DRILLS.

Economy in drilling depends to a great extent on the ability of the blacksmith to properly sharpen and temper drills. If the drills are too hard they will chip easily, and if too soft they dull rapidly. Uniformity in the size of drills is also important. Much annoyance and loss of time will probably result from an endeavor to continue a drill hole with a drill slightly larger than the one employed to drill the first part of the hole.

WEDGING.

TYPES OF WEDGES EMPLOYED.

In some quarries wedges that reach only a short distance into the drill holes are employed. As the entire strain is near the rock surface heavy sledging is necessary, and uneven fractures may result. The method is not effective.

The long wedges commonly used represent a marked improvement. Iron plates or feathers are attached to their extremities with wire in order to hold them in proper position while being inserted into the drill holes. When the wedges are driven the strain is thus exerted at points near the bottoms of the drill holes. If alternate holes are made half depth the strain is more evenly distributed.

It has been mentioned in a previous paragraph that one quarry company reduced the size of drill holes to 1⅛ inches. Ordinary wedges reduced to fit such holes were not strong enough to withstand the heavy sledging required. Consequently a wedge was designed that gave effective service with lighter blows.

The type that has proven highly successful is shown in figure 13 The feathers are 3 feet long and the plug 3 feet 9 inches, the additional

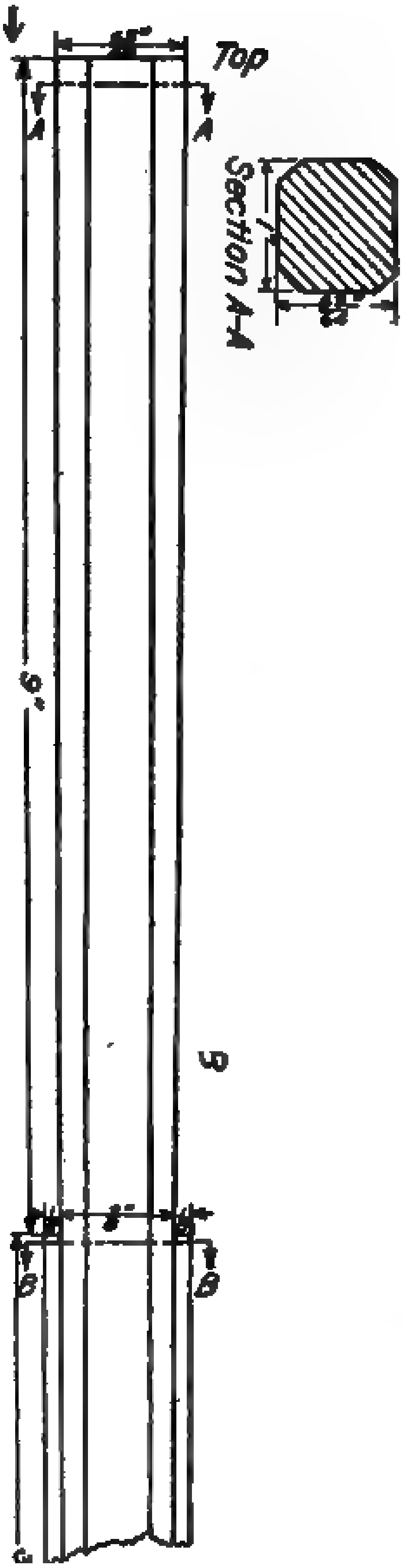

FIGURE 13.—Details of wedge that is effective in a small drill hole.

9 inches being required for driving. The plug tapers gradually from $\frac{3}{4}$ inch in diameter at the top to $\frac{7}{16}$ inch at the bottom. The feathers are curved on one surface to fit the drill hole, and the flat surface is perfectly straight and gives a uniform taper from one end to the other. They are made true to form in a swage. The important feature of the design is that the diameter B–B at the top of the feathers is exactly the same as the diameter C–C at the bottom, $1\frac{1}{8}$ inches. As a consequence when the plug and the feathers are inserted into the drill hole in the position shown in the figure, one side of each feather is in contact with the plug, and the other side with the wall of the drill hole throughout its entire length. When the plug is driven the feathers are forced apart a uniform distance at every point, and as a result the pressure exerted is uniformly distributed over their full length. Straight and even fractures result, and are obtained with much lighter sledging than by any other method yet devised.

UNIFORM STRAIN DESIRED.

It is important that in driving wedges there should be equal strain on all of them. If wedges near one end of a long mass of marble are driven much harder than those near the other end, an irregular cross break may result. In other words an artificial strain break may be produced, especially in long breaks in marble that has no rift. Wedging such marble should never be unduly hastened. A more uniform break will result by giving the rock time to fracture gradually than by forcing it. Uniformity of strain and moderation in the rate of fracturing may both be best accomplished by em-

ploying one man only on a single floor break. Different men strike blows with different degrees of force, and thus although one man may drive a wedge with the same number of blows as another, one may exert a much greater strain than the other. Also when two or more men are employed, there is a tendency to force the break at a too rapid rate.

If short breaks are made, or if the marble has a decided rift parallel with the direction in which fracture is desired, more latitude may be allowed. In some quarries several men work together in driving wedges, a leader giving the word so that all strike at once. A gang may, with practice, work together in this manner and produce a uniform strain. If the rapid splitting has no ill effect the method is justified.

EFFECT OF RIFT ON DRILLING AND WEDGING.

It has already been stated that the process of splitting is greatly facilitated by rift. If there is an exceptionally pronounced rift in a horizontal direction, it may be possible to make a floor break double the width of an ordinary marble block. This method is employed by a Maryland company. After the wall cuts have been completed, channel cuts are made 12 feet apart and in one direction only. A floor break is made the full 12 feet wide, and then the mass of marble is split lengthwise by drilling and wedging. Thus, in one direction, the rock is intersected by alternate channel cuts and rows of drill holes, and all the cross breaks are made by wedging. As the cost of drilling and wedging is less than that of channeling per square foot of surface produced, a considerable saving is effected.

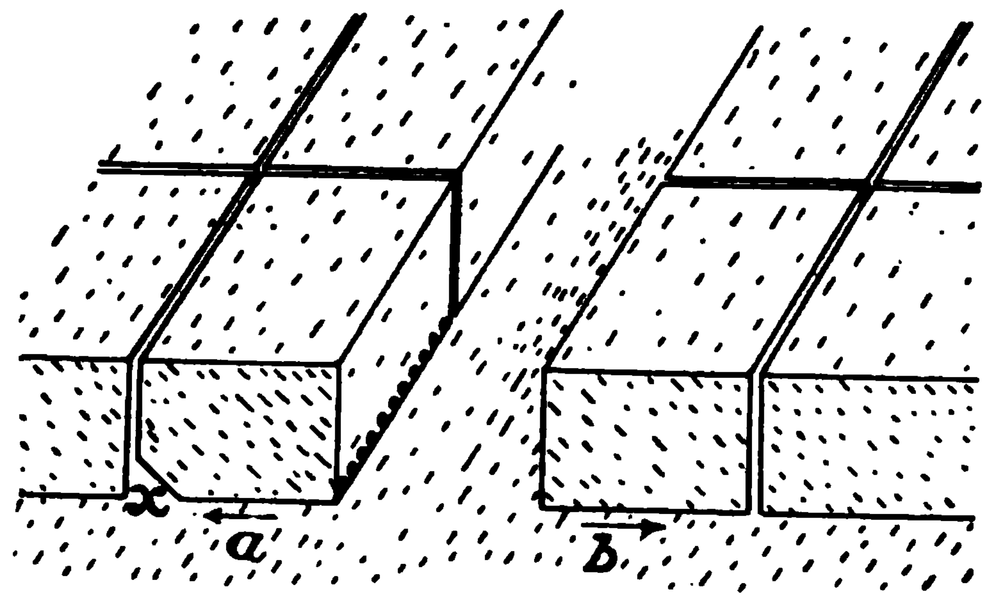

FIGURE 14.—Influence of rift on bottom breaks. Holes drilled in direction *a*, if too shallow, will result in break at *x* and injure blocks; holes drilled in direction *b* will result in a straighter break.

In most marble there is a decided rift parallel with the bedding. Therefore, if the bedding dips at a steep angle, the rift may be inclined in like manner. If the rift is inclined and the quarry floor level, the direction in which drill holes are projected for floor breaks is a matter of considerable importance. In a Colorado quarry in which the floor is level and the rift steeply inclined the channel cuts are made parallel with the strike of the rock. The allowance that should be made for the influence of rift on the process of wedging under such conditions is shown in figure 14. When the row of key blocks has been removed holes may be drilled in the direction shown by the arrow *a* in the

figure. When the holes are wedged there is a tendency for the break to leave the plane of the drill holes and slant upward on the rift, thus removing a corner of the block, as at *x*. When the holes are drilled in the opposite direction, shown by the arrow *b*, if the channel cut is not continued lower than the plane of the drill holes, the break will be straight, as it will not run down below the bottom of the channel cut. As a consequence, when the holes are drilled in the direction *a* they must be made deeper than when drilled in the opposite direction. It is apparent that, to avoid waste by broken corners and to reduce the expense of drilling, the row of key blocks should be taken out as near as possible to the left side of the quarry as viewed in the figure in order that most of the drilling may be done in the direction *b*.

In cutting out masses of marble with the wire saw in this quarry it was difficult to maintain a uniform quarry floor. The saw cuts were projected downward 18 or 20 feet. In making the floor breaks in the direction *a* (fig. 14) there was a tendency for the break to pass upward and remove a corner of the block, and in making the floor break in the opposite direction the break would slant downward on the rift and remove a corner of a block on the floor beneath.

REMOVAL OF KEY BLOCKS.

Various methods are employed for removal of key blocks. As in many other quarry operations, the method is controlled to some extent by the conditions and to some extent by the skill and experience of the foreman. The key block may be removed in fragments as waste, it may be divided into a number of small blocks, or it may be removed in its entirety.

REMOVAL OF KEY BLOCKS AS WASTE.

If a band or mass of poorer material traverses the quarry, the key blocks may be located in it. The mass may be blasted into fragments and removed as waste. This method is employed by a North Carolina company. A narrow band of flinty inferior material passes through the middle of the quarry. The key blocks, consisting of this flinty material, are removed by blasting, and no unnecessary loss ensues.

However, some quarry experts think that even inferior rock can be removed more cheaply in large masses with the derrick than by breaking it and removing it in small fragments. In areas beyond the reach of the derrick the latter method is to be preferred.

Even if no such inferior bands or masses occur, some quarrymen blast and remove the first key block as waste, believing this method to be the most rapid and convenient. Unless the marble thus quarried is of low grade, such economy of time will not justify the waste of material.

MAKING THE FLOOR BREAK.

When the key block is to be preserved, the first step after channeling has been completed is to make the floor break for the first block. A common method is to insert a slanting iron plate in the bottom of the channel cut and to place the point of a wedge between it and the the key block. When the wedge is driven, the entire strain is exerted at the bottom of the block. A series of such wedges may be placed close together. If the floor is intersected by other parallel channel cuts, it is necessary to place wedges in those immediately behind the point where wedging is done, to insure that the fracture will take place where desired.

A method now rarely used is to charge an iron tube with blasting powder, tamping it hard above the charge and plugging it lightly below. The end of the pipe containing the charge is placed in the bottom of the channel cut and a strain put on the block by driving wedges. When the charge is fired, a clean break is said to result from the shock. A small charge placed in a hole drilled in the center of the block the exact depth of the channel cuts will act in much the same manner. Such a hole may, however, constitute a serious defect in the block.

After the first block has been removed, horizontal holes may be drilled and the next block broken loose by wedging in the ordinary manner. When the working space is too confined for a bar drill, a jackhammer may be used. When the bench has an open side, the first block may be drilled by placing a drill bar in a waste box and suspending it in proper position with the derrick.

A special method is employed in a Vermont quarry, where operations are conducted on a floor slanting about 45°. If key blocks were drilled and wedged in the ordinary manner, there would be great danger of the blocks sliding down upon the men the moment they were broken loose. To overcome this danger, a single drill hole is projected at the center of the floor line, and a light powder charge is exploded in it. The charge is so small that it makes the floor break without otherwise shattering the block.

HOISTING OUT KEY BLOCKS.

A method that is too wasteful to be recommended is to quarter the first key block with the channeling machine and remove it in sections. Removal in four pieces is undoubtedly easier than in one. There are two serious objections, however—first, the additional expense of making the two extra channel cuts, and, second, the decreased value of the material in small blocks. It is probable that the marble in four pieces is worth less than half as much as the intact block.

There are now in common use three methods of removing the first key block in its entirety. The first of these is by the use of the Lewis pin, which is adapted only to strong rock. A hole several inches deep is drilled into the middle of the block. A bar with an eye in the top is placed in the hole with a wedge at each side of it, as shown in figure 15. The bar is thicker at the bottom than at the top, so that when pulled upward it tends to tighten on the wedges, when the block may be lifted out with the derrick hoist.

A second method which may also be employed in strong rock is the use of grabhooks. Small pieces may have to be broken from the corners of adjoining blocks in order that the holes may be drilled properly and that there may be room for the hooks.

A third method is employed if the beds are weak. Chain loops or cables are thrown over the block from opposite sides. They are placed near the bottom of the block and are drawn tight and the block is lifted out with the derrick hoist.

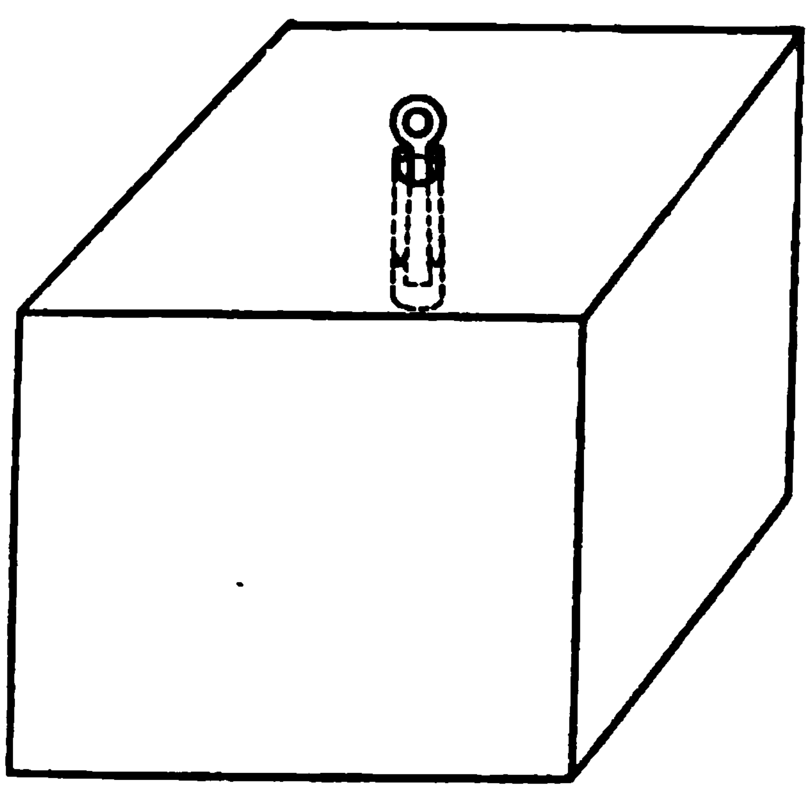

FIGURE 15.—Lewis pin used in hoisting key blocks.

HOISTING.

TURNING DOWN BLOCKS.

Blocks are usually turned down with a quarry bar before the hoist cable is attached. When monoclinic blocks are excavated it is difficult to turn them down. In a Vermont quarry the tip of the block is raised with a bar and a cable sling placed beneath it. The block is then raised several inches with the derrick and blocked up ready for final cable attachment.

The employment of the derrick to assist the bar gang in turning down blocks may result in serious risks to those employed. After the first fracture has been made a block is by no means free. The interlocking of irregularities on the surface may require considerable strain before the block will move. When the derrick is employed there is grave danger of breaking a boom or cable with consequent danger of serious accident.

CABLE ATTACHMENT.

The hoist cable may be attached to blocks by grabhooks or chain or cable slings. Grabhooks are employed only with rock that is hard and coherent. The two holes for the hooks are made on opposite sides of the block a few inches from the top. A mistake is

sometimes made in drilling the grabhook holes too deep. The chief strain then comes, not at the tips of the hooks, but on the curved parts that are in contact with the upper edge of the block. As a consequence a corner of the block may chip off and allow the whole mass to fall. The holes should be sufficiently deep to allow a firm grip of the rock but the chief pressure should fall on the tip of the hook in the bottom of the hole. Moreover, great care must be taken in hoisting by this method. The rock should be carefully balanced, as a partial rotation may cause the hooks to slip. Under no conditions whatever should a quarry workman occupy a position beneath a block that is being lifted with grabhooks.

A safer method of attachment is to pass chains under the block and completely around it. Blocks are held much more securely by this method and there is little danger of a mass of rock falling on account of a weak bedding plane or fracture. A chain is, however, rather uncertain in its strength, and detection of weak links is somewhat difficult. When a chain does break the accident is usually quite unexpected. A wide margin of safety and frequent examination are necessary.

The best method of cable attachment is by means of cable slings (Pl. VI, *A*). Such slings are quickly handled and the use of two slings renders balancing of the block comparatively easy. Defects in steel cables are easily recognized. A realization that slings are both safe and convenient has led to their adoption in many quarries.

EFFICIENT HANDLING OF MATERIAL.

All unnecessary handling of material should be avoided. Blocks should be hoisted from the quarry and loaded on cars at a single operation if possible. If circumstances will not permit this, they should be placed in a convenient position for future loading. At most quarries hoisting and loading are done in a fairly efficient manner.

However, more efficient methods of handling waste material should be devised. In many quarries there is great loss of time and power in rehandling waste blocks. At one quarry waste blocks were hoisted from a point near the surface at one side of an excavation, thrown to the bottom of a 50-foot pit, picked up with a second derrick, and deposited on cars. Cars were situated within easy reach of the first derrick, but the placing of the blocks on them necessitated swinging the blocks over channeling machines. Usually such conflict of operations may be avoided by careful and judicious plans for quarry development. Every additional operation in handling waste adds to the cost per cubic foot, and every increase in such cost cuts down the margin of profit on the finished product.

TUNNELING.

DEFINITION OF TUNNEL.

In quarrying the term "tunnel" is applied to a subterranean working, level or inclined, having a roof of undisturbed rock. The term is used in contrast with the "open-pit" quarry in which the opening is the full size of the excavation, and with the "undercut" quarry, the walls of which slant so as to make the floor space wider with increasing depth.

GENERAL CONSIDERATIONS.

Although open-pit quarrying is the more common type, there are conditions under which tunnel methods are to be preferred. A tunnel affords a means of quarrying out desirable beds without the removal of heavy stripping. However, there are certain difficulties and dangers that must be carefully considered. Among them may be mentioned the danger of roof falls, the expense of artificial light and ventilation, and the necessity of additional handling of quarry material. A necessary condition for successful tunneling is the presence of a strong roof. In regions where faulting or jointing are pronounced the danger from roof falls may be so great that tunnel methods are impracticable.

OPENING TUNNELS ACROSS BEDS.

Procedure to be followed in tunneling a deposit in which there are no open bedding planes or when the tunnel roof is to cross the beds may be illustrated by describing the method used in a Vermont quarry. A preliminary opening 6 to 7 feet high is made by channeling and blasting. As the rock thus removed is waste, the opening is, if possible, made in inferior material. Determination is made that the thickness of sound rock above the tunnel is such that there is no danger of roof collapse. After the proper position for the ceiling has been chosen a channeling machine is used to cut a channel about 7 feet deep, starting about 3 feet from the floor and slanting downward to meet the floor line. A row of horizontal drill holes is then made at the roof and another row is made parallel with the floor of this preliminary tunnel. In addition vertical rows of holes are driven at intervals of about 7 feet. The relative positions of channel cuts and drill holes are shown in figure 16. The lower wedge-shaped mass of rock x in the figure is dislodged by blasting in the drill holes below the channel cut. The upper overhanging ledge y is then broken down by discharging blasts in the holes above the channel cut. Black blasting powder is used and all charges are exploded by fuse. Such blasting is probably less effective than a simultaneous discharge of

A. CABLE SLING FOR HOISTING BLOCKS OF STONE.

B. UNDERCUTTING PARALLEL WITH STEEPLY INCLINED BEDS IN QUARRY AT WEST STOCKBRIDGE, MASS.

C. A BALL BREAKER USED IN QUARRY AT GANTTS QUARRY, ALA.

blasts by means of electric detonators. However, if the tunnel is driven in tight beds or if it crosses the beds the simultaneous discharge of a number of blasts would undoubtedly shatter and thus destroy marble adjacent to the tunnel.

When, by repeated blasting, the mass of rock 6 or 7 feet in depth has been shattered across the entire width of the tunnel, the next step is to remove the waste material. Large pans holding 2 to 6 tons each of rock are used. Each pan is shaped like a sugar scoop and is handled by means of a cable fastened to the back of the pan, passing to a second attachment at the front, and then over a sheave in the tunnel roof near the edge of the excavation. Rock fragments are loaded in by hand and the pan is hauled to the edge of the shelf. The cable is then released from the front of the pan, and further hauling causes the back of the pan to be raised, allowing the contents to slide out. If the tunnel floor slants downward from front to back, the empty pan is returned on rollers, whereas if the floor is level or inclined upward the pan is hauled back by means of a cable attached to the back and passed around a sheave at the extremity of the tunnel. The rock fragments are again loaded into pans at the base of the shelf, hauled by trolley or cable cars to the main hoist, elevated to the surface, and loaded directly on railroad cars with the derrick.

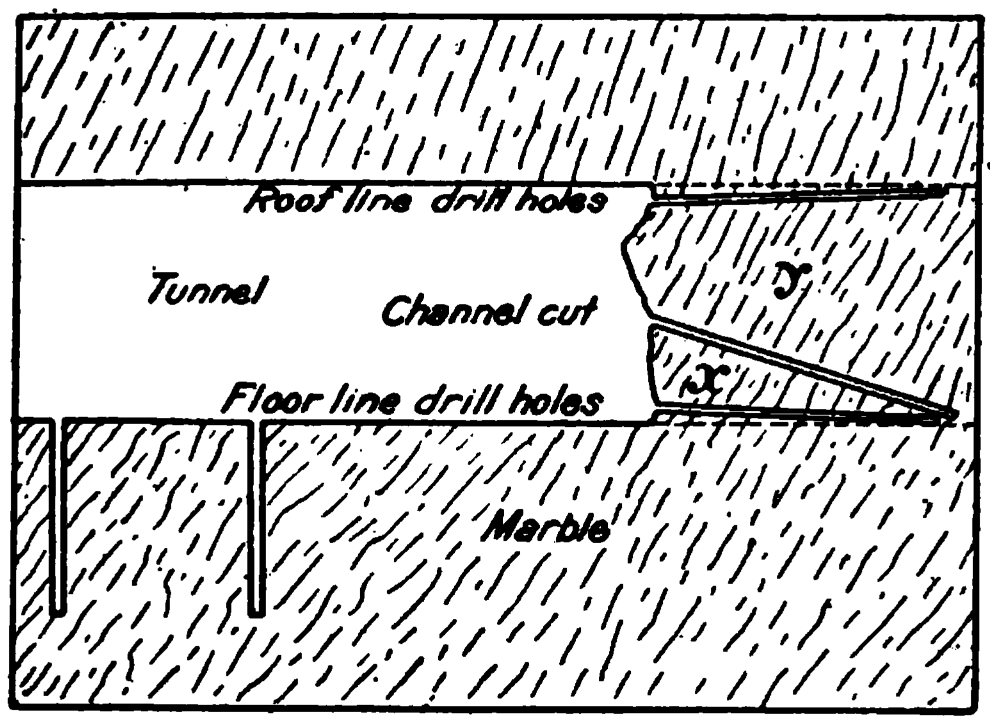

FIGURE 16.—Method of driving a tunnel in marble.

The process of channeling, drilling, blasting, and removal of fragments is repeated over and over until the preliminary tunnel has been projected far enough to give sufficient floor space for economical operations. Plate VII, *A*, shows the appearance of a tunnel in one of the Vermont quarries.

METHOD OF OPENING TUNNELS PARALLEL WITH OPEN BEDS.

In an Alabama quarry tunnels are driven parallel with the beds, which dip about 33°. The tunnel floor is made coincident with a band of soft mica schist which occupies a position between two marble beds. The soft band is thus utilized to take the place of the channel cut as described in the preceding method. One expensive operation is thus saved. Drill holes are projected in a row parallel with the roof, and vertical rows of holes pass from the

roof to the floor at intervals. Black blasting powder shots are discharged in simultaneous groups by an electric firing machine. The presence of the band of mica schist at the floor acts as a cushion and prevents the shock from shattering the good marble below. On this account, the more effective simultaneous blasting may be employed.

In removing the material, loaded buckets holding 4 cubic yards are hauled to the edge of the excavation but are not dumped. By means of a derrick at the mouth of the tunnel they are transferred to a position from which they may be hoisted with the main derrick and dumped into railroad cars.

METHOD OF QUARRYING ON TUNNEL FLOOR.

When a preliminary tunnel of sufficient size has been completed, channeling machines may be set up on the floor and operated as in an open quarry. Electricity is the most convenient power for use in tunnels. Air-driven machines are better than steam-driven, as the space for operation is usually confined. The channelers may work on a level or an inclined floor. At West Rutland, Vt., the upper levels are quarried on a horizontal floor and the lower ones on an inclined floor. In an Alabama quarry the channel floor is inclined and the channel cuts intersect the slanting floor diagonally.

ROOF SUPPORTS.

In a West Rutland, Vt., quarry the tunnel roofs are supported by pillars of marble each 20 feet square left at various intervals. The spacing of the pillars is governed by the evident security of the roof. If the beds are heavy and seams absent, the spacing may attain a maximum of 100 feet. In sections where the beds are seemingly less secure the pillars are spaced at intervals of 60 to 80 feet. In early days if the roof slanted the pillars were inclined to form right angles with the roof. Inclined pillars, however, occupy more space, and they form more serious obstacles to the operation of derricks than vertical pillars. Of late years all the pillars are made vertical though the roof may be inclined.

In an Alabama quarry a wall running in the direction of the dip is left, rather than a series of pillars. This plan was adopted after due consideration of the possibility of pillars slipping on a soft, inclined, interbedded mica schist.

Figure 17 illustrates the present and proposed future arrangement of pillars in a Colorado quarry. The tendency to stagger alternate rows is noteworthy.

A. A TUNNEL IN A VERMONT MARBLE QUARRY.

B. GANTRY CRANE EMPLOYED AT QUARRY AT ASHLEY FALLS, MASS.

C. MANNER IN WHICH OPEN SEAMS PITCH INTO THE CORNER OF A QUARRY WHERE THE QUARRY WALL MEETS THE STRIKE OBLIQUELY.

TRANSPORTATION OF MATERIAL IN TUNNELS.

Blocks of marble or boxes of waste material may be hauled out of tunnels by means of cables. Quarry cars operating on tracks are more efficient. One company operates both cable cars and an electric trolley. If an efficient means of transportation is not provided, great loss of time will result.

LIGHTING AND VENTILATION OF TUNNELS.

In order to promote safety and efficiency in tunnels, adequate lighting is necessary. Either arc or powerful incandescent lights are suitable. If blasting is employed, the harmful gases should be

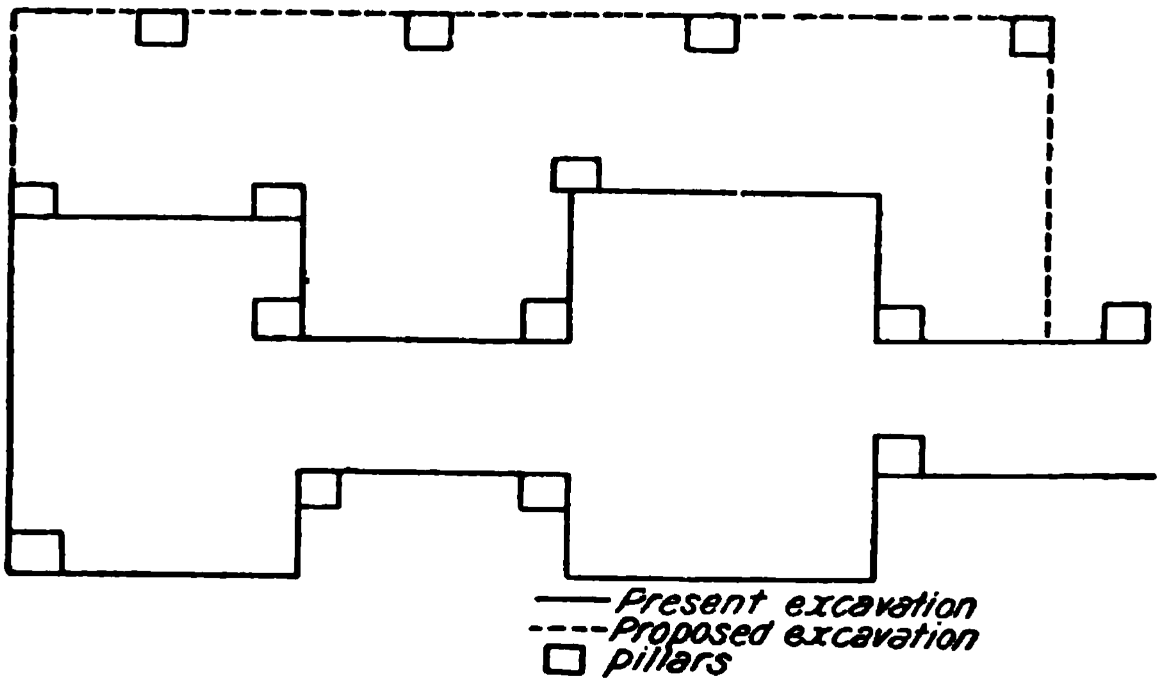

FIGURE 17.—Present and proposed arrangement of pillars in a Colorado quarry.

removed. In the ventilation of a Vermont quarry large wooden conduits through which the impure air is drawn by powerful fans are used.

UNDERCUTTING.

A modification of the tunnel method consists in enlarging the quarry floor by an outward inclination of the wall cuts. The process is simple, requiring no additional equipment and no expensive preliminary operation. A wide floor space is obtained with a minimum of stripping, and with moderate extension no supporting pillars are present to obstruct quarry operations. There are, however, certain disadvantages. In tunneling, the projection of a preliminary opening is costly and produces only waste rock, but when once completed the subsequent operations of channeling and drilling are carried on with almost the same facility as in an open quarry. In undercutting, however, every wall cut is slanting. Channeling at an angle is slow and relatively expensive. Also the blocks of the outer row are angular in shape, resulting in waste.

In extensive undercutting, the danger from overhanging rock may be averted by leaving wing supports of marble at intervals. Under-

cutting is employed successfully in many of the Georgia marble quarries. It is to be recommended if the rock is sound, and if tunnel methods are too expensive or for other reasons seem impracticable.

In a Massachusetts quarry the walls are undercut parallel with the steeply inclined beds as illustrated in Plate VI, *B*. By this means the desirable beds are followed and the floor space gradually increased. The blocks of the first row are removed as keys and subsequent channels are cut at right angles to the strike.

COMBINED TUNNEL AND UNDERCUT.

If a band of marble is vertical in attitude or inclined at a steep angle and not exceeding 150 or 200 feet in thickness, a safe and economical method of extraction is to project a narrow tunnel and gradually widen it by undercutting. If through weathering the marble has been altered to a considerable depth, it may be possible to project the preliminary tunnel in inferior rock. Undercutting will gradually widen the floor, and the steep walls thus formed are less liable to scale than a flat roof. Undercutting is not to be recommended, except when the lateral walls are to be permanent, for as a rule, it is not practicable to further extend undercut walls. When the lateral limits of the desirable material have been reached, the angle of inclination of the wall cuts may be modified to conform with the dip of the marble beds.

DRAINAGE.

There is great variation in the quantity of water encountered in quarries. In some quarries heavy springs are encountered which require constant pumping; in others surface drainage only must be removed. Steam or motor driven reciprocating pumps are most commonly employed, though motor-driven centrifugal pumps have been used successfully, and are highly recommended by a number of quarry operators.

GENERAL CONSIDERATIONS IN QUARRYING.

Whenever possible, it is wise to conduct quarry operations on a wide floor space. In general, operations on a large scale are more economical than those on a small scale. A large floor space also allows greater choice of material. The encountering of local unsoundness may temporarily suspend production in a small quarry, whereas in a larger one the unsound rock may be avoided for the time being and removed during a dull season.

Greater efficiency in quarry operations may be attained by employing night shifts. By this means the output can be almost doubled without any increase in the power plant or quarry equip-

ment over that required for the day shift only. The work should be so arranged that the night shift is relieved of those operations that on account of poor light, or for other reasons, are not performed to advantage in the night. Machine work is the most satisfactory of all operations for night shifts, and consequently, the day shift should endeavor to leave the machines in the best possible condition for continuous operation.

All quarries should be provided with safe and efficient stairs or ladders.[a] Much time is lost when men are obliged to pick their way over steep and rough rock surfaces or climb down hastily improvised steps or ladders.

Removal of mud and rock fragments from the quarry floor is an operation that should not be neglected. An accumulation of débris impedes quarry operations. The presence of a large heap of rubbish in one part of a quarry may discourage the proper performance of a task and lead to the substitution of a less-efficient method. All rubbish should be cleaned up systematically and frequently, and the entire floor space kept in a condition favoring a free and intelligent performance of all subsequent tasks.

SAFETY FIRST.

"Safety first" should be the motto in planning and erecting all quarry equipment and in the conduct of all operations[b] Many operators fail to realize that safety devices contribute to efficiency. They consider that much of the money spent in safeguarding machinery and the time employed in making examination for possible dangers is entirely lost. The matter of safety should be viewed in a different light. Men become greater producers if they feel that all reasonable means are employed to safeguard their lives and health. Much time is lost if men feel that they must constantly be on the lookout for some overhead dangers. Moreover, accidents deprive the quarry of the services of skilled men and frequently burden the proprietor with a load of expense.

HOISTING EQUIPMENT.

TYPES OF DERRICKS.

Various types of derricks are employed in marble quarries. Both the mast and the boom may be of wood, the mast may be of wood and the boom of steel, or both may be of steel. Spliced wooden derricks having mast and boom each in four pieces are used in some regions. They are strongly supported with iron bars and turnbuckles. They are easy to transport and set up. Those in common use

[a] See Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, pp. 15-16.
[b] See Bowles, Oliver, Op. cit., pp. 12-36.

have a lifting capacity of 15 to 18 tons. All derricks should be painted to protect them from the weather.

USE OF GANTRY CRANES.

A Massachusetts marble company uses a traveling gantry crane in place of a derrick, as illustrated in Plate VII, *B*. It has an 80-foot span and a lifting capacity of 25 tons. The entire crane travels back and forth on tracks placed at each side of the excavation. It is readily accessible to all parts of the quarry, and blocks once lifted are transported directly to the stock pile without further handling. One disadvantage is that the use of the gantry crane limits the width of the quarry. Furthermore, it is not to be recommended for quarries in the development of which considerable hoisting will be required, as it necessitates that all lifting be done with one hoist. If derricks are employed several may be placed in convenient locations and operated simultaneously.

DERRICK GUYS.

All derricks should be strongly supported. For 20-ton derricks eight iron guys of 1¼-inch diameter are necessary. For derricks with 90 to 100 foot masts and with a maximum lifting capacity of 35 tons the guys should be 1¼-inch steel. For locking guys, Crosby, Roebling, or Leschen clips are regarded as more reliable than splices.

A new arrangement of guys is now employed in some quarries in Vermont. Instead of 12 guys being spaced regularly, they are arranged in four sets of three each, the groups radiating at successive angles of 90° from the mast. An advantage lies in the fact that four large secure concrete piers may be constructed at a smaller cost than 12 small and possibly unsafe ones. Where several derricks are situated close together, it is often a problem to properly place guys and at the same time evenly distribute the strain. The arrangement mentioned overcomes this difficulty in a most satisfactory manner.

Where several derricks are situated near together, the top of one mast is occasionally anchored to the top of another. There is considerable risk in this method, as strains on different derricks may occur in the same direction simultaneously and thus multiply the strain on a single guy.

GUY ANCHORS.

Guy anchors are of various sorts. Where bedrock outcrops at the surface the safest method is to drill holes in the rock, insert a bar with a ring in the top, and fill up the hole with melted sulphur. Where considerable depth of soil surrounds the derrick, guys may be anchored to buried timbers. The trunks of large trees, preferably grow-

ing trees, are sure supports. It is advisable, however, to place a secondary anchor to support the tree.

A post supported by a pile of stones, as shown in Plate VIII, *A*, is used by some Tennessee quarrymen. The post should be nearly vertical. It should not be slanted backwards, as in that position it forms a lever tending to raise the soil or rock in front of it, and thus is not in the best position to withstand strain. Such posts should also be supplied with secondary anchors.

A concrete pier in which is embedded an angle-steel bar is a highly efficient support. An eyebolt may be passed through the bar and the slack of the guy taken up with a nut, as shown in Plate VIII, *B*.

HOISTS.

Steam hoists are the type most commonly employed in marble quarries, although compressed-air and electric hoists are used in several places. The type of hoist to be employed depends on the nature of the work. Deep quarries demand a higher cable velocity than shallow ones. In any case a very slow motion is not desirable. Much time is lost in quarries where men must wait while blocks are slowly raised to the surface, especially if rock excavation is being carried on near the point of elevation, as safety demands that men must move from their working places while the block is overhead. The power required for hoisting increases rapidly as the speed is increased, and a proper balance must therefore be maintained between the speed of hoisting and the power consumed for the operation.

In many quarries crude methods of sluing are still employed. Sluing is sometimes done by means of a rope attached to the block or by a mechanical sluing device operated by a handwheel. The sluing device that forms a part of all modern hoists is much more efficient, as it gives a more rapid motion than hand-operated devices and is controlled by the hoist engineer.

The spinning of blocks while suspended endangers the lives of employees and makes difficult the guidance of their course during ascent. Spinning may be avoided by using a nonspinning hoisting cable. The central strands of such a cable are twisted to the left and the outer strands are twisted to the right.

LOCATION AND SIZE OF DERRICKS.

The quarryman should have a clear and definite plan of quarry development before placing his derrick. The extension of a quarry excavation in a direction away from the derrick may soon bring it beyond the range of the boom. Thus the erection of a new derrick,

and possibly a second handling of material, may be necessary. In several quarries blocks are hauled long distances by cables before they reach a position from which they may be hoisted to the surface. Such preliminary haulage is injurious to cables, involves great loss of time, seriously interferes with other quarry operations, and is surrounded by many dangers. It may be advisable to install a car-haulage system or to replace or supplement a derrick by a new and larger one. The size of the derrick must also be governed by the extent and direction of quarry operations. A small derrick may give satisfactory service for a few months, after which it may have to be replaced by a larger one. It is usually more economical to erect a large and permanent derrick at first, provided the operator is reasonably certain that quarrying within its range is to be extensive.

HOIST SIGNALING.

In shallow quarries or in yard operations signaling by hand motion is almost universal. If the hoist engineer can not see the bottom of the quarry, hand-motion signals are sometimes relayed to a man in an intermediate position. Such a method is not to be recommended, as the repetition of signals involves increased danger of misunderstanding and also requires an additional man.

Types of mechanical signal apparatus in common use are electric bells, telephones, and bell pulls. Dry-cell electric bells are not considered reliable; they may fail to act at critical times. Electric bells operated by wet-cell batteries are more reliable if inspected at regular intervals. Telephone connections require inconvenient ear attachments. If noisy operations are conducted, words given by telephone may be easily misunderstood. The bell pull is considered to be the most convenient and reliable method. Both the apparatus and its operation are simple and there is little danger of misunderstanding signals.

SCABBLING.

Where the mill is situated close to the quarry, the process of scabbling may be omitted. If situated at a distance, or if the marble is to be sold in block form, the blocks should be scabbled to avoid transportation of waste material. The most common method of scabbling marble blocks is by manual labor with the scabbling pick. The wire saw is used successfully in some places. Hammer drills and wedges are used occasionally for the removal of the more prominent surface irregularities.

A. A GUY ANCHOR POST USED IN TENNESSEE

B. AN ANGLE STEEL AND CONCRETE GUY ANCHOR USED IN A QUARRY AT GANTTS QUARRY, ALA.

TRANSPORTATION OF QUARRIED ROCK TO THE MILL.

VARIOUS METHODS OF HAULAGE EMPLOYED.

In some quarry regions mills are so favorably situated that short haulage only is required. In several eastern localities blocks are loaded directly upon the transfer cars by means of the quarry derrick. A number of companies use railroad cars and locomotives. Electric trolley lines are used successfully even on heavy grades. On such grades efficient and dependable brakes must be used. Steam tractors may be employed if roads are graded and firm. It is a matter of considerable expense to grade and keep in proper repair roads suitable for the heavy traffic incident to the use of tractors.

TEAMS AND WAGONS.

Teams and wagons are used in many places. Where quarries are situated on mountain sides and rock must be hauled down steep roads, the chain or shoe attached to the wheel to prevent too rapid descent is destructive to roads. Haulage by horse-drawn vehicles entails a slow rate of haulage, small loads, much repairing of roads, and heavy cost for maintaining horses. Hence, in many places, some form of cable-car transportation would seem to be more suitable than by teams and wagons. Where light grades are encountered, mules or horses may be employed to haul cars.

CONNECTING QUARRIES WITH RAILROAD TRACKS.

Many marble quarries are situated at short distances from railroads, and some form of conveyance is necessary to connect the quarry with the railroad track. Where the grade is light, a railroad siding is to be preferred. Where small quarry cable cars are used, it becomes necessary to erect a derrick at the junction point and rehandle all marble going out and all coal coming in. Where a siding is built, transportation may be conducted by means of standard railroad cars with no transshipping. Where a heavy grade is encountered, cable cars are necessary and transshipping may be unavoidable.

CABLE CARS.

Cable cars are used successfully in a number of regions. The conditions vary so greatly that no one particular form can be recommended. The system must be modified to suit the conditions.

One of the most complete cable-car systems in use in any American marble quarry is that in a Vermont quarry. Two cars are used, the empty car ascending while the loaded car descends. The hoist is operated by a 75-horsepower electric motor. A $1\frac{1}{2}$-inch

steel cable passes from one car to the hoist where it makes six turns on a 12-foot drum, then passes out and is attached to the second car. The cable is guided by rollers. The track is three-fourths of a mile long. Over part of its course the grade is light, but at two points it is very steep. The cars pass at the center. Above this point a three-rail track is provided to prevent interference of the ascending and descending parts of the cable. Below the center the cable is always single, and in consequence a single track only is necessary. The maximum load is 40 tons, and 20 minutes is required for each trip. By means of an automatic needle indicator, which travels back and forth on a board, the hoist engineer can accurately follow the course of the loaded car. The rock is transferred from the cable cars to standard railroad cars for transportation to the various mills.

FACTORS CONTROLLING THE METHOD OF TRANSPORTATION.

Before a system of transportation for any particular quarry is adopted the conditions must be studied carefully. The most important considerations are probable output, size of material, number of times it must be handled, rate of haulage, distance, surface conformity of the country to be traversed, first cost of the system, cost of maintenance of equipment and roadway, and capital available for transportation purposes.

EQUIPMENT AND OPERATIONS IN MILLS AND SHOPS.

At least one-half the marble quarries of the United States have more or less completely equipped plants for sawing, polishing, carving, or otherwise preparing marble for structural uses. The operation and equipment in many such mills and shops were noted in connection with quarry observations. Marble mills and finishing plants belong to the manufacturing rather than to the producing phase of the marble industry, and consequently may not strictly be included in a discussion of the technology of marble quarrying. The brief discussion of marble-manufacturing plants presented in this section is justified by the direct interest that many quarrymen have in the manufacturing phase of the industry. Although certain facts relating to method and equipment are presented, for whatever immediate benefits they may suggest, no attempt is made to deal with the subject exhaustively at this time.

LOCATION OF PLANT.

The location of marble works is influenced greatly by water supply and availability of power. Aside from these considerations, although in many respects it is more convenient to erect the plant near the quarry, conditions may be such that this plan is unwise. If the

quarry is situated several miles from towns, in a region where roads are poor, it may be wiser to build the plant at some town, especially if other marble plants are centered there.

There are several arguments in favor of placing stone manufacturing plants at a common center rather than at the quarries. Although such an arrangement requires transportation of rock that is later wasted in saw cuts and coping, buyers can the more readily conduct their business if shops are centralized. Furthermore, it is easier to keep men near towns where schools and other public institutions are more convenient and more efficient than in comparatively unsettled regions. If shops are situated at a considerable distance from railway lines, sidings are often uneven and transportation over them is more destructive to thin-finished material than to uncut blocks.

BUILDINGS.

Fireproof mills and shops are to be preferred. When a fire occurs the loss is great, because the heat is liable to destroy all the machinery. If buildings are not already of fireproof construction, a sprinkler system is to be recommended. One advantage of such a system is that water can do little damage. It may possibly cause swelling of the timbers, thus throwing the machinery out of alignment, but otherwise most of the equipment is of such a nature that water will not affect it.

The plan of shop and mill construction carried out by a Georgia marble company is worthy of remark. The company constructed with brick and concrete an absolutely fireproof section separating two units that were not fireproof. With such an arrangement fire can not paralyze production, because if one unit is destroyed the fireproof section prevents spreading of the fire, and operations may be continued in the remaining unit.

HEATING AND VENTILATION.

In cold climates the necessity of preventing the freezing of water employed in the various processes, and also the need of keeping men comfortable while at work, demands that shops be closed structures. A system of heating and ventilation also becomes necessary. Coal stoves are employed in many mills and shops, but are not satisfactory, especially in large buildings. They require too much attention, increase the fire risk, and provide an unequal distribution of heat. In a Vermont mill the air, impelled by powerful fans, is passed through coils over a furnace, and conducted through overhead pipes, from which it is expelled downward at various points. With adequate renewal of circulating air from outside sources, heating and ventilation may be combined into one system. A Pennsylvania company uses the exhaust steam from its power plant to heat the mill and shop.

POWER.

Direct water power is highly satisfactory. The cost of operation and maintenance of turbines is relatively low. An adequate water supply at all seasons of the year is desirable in order that continuous operation may not be interrupted through lack of power. At one Vermont mill two turbines on vertical shafts supply power to an overhead countershaft by means of bevel gears. The turbines are so arranged that one can be disconnected at night or at such other times as only partial power is required.

Steam is a common source of power for mills and shops. Where fuel is cheap steam is satisfactory.

Electricity, which may be developed by the company or purchased from power lines, is used in many places. Where an abundant supply and a good head of water may be obtained with a small outlay of capital, hydroelectric power may be cheaper than steam even though the cost of coal is low. The convenience in transmission of electricity has induced some companies to develop it by steam in places where there is not a sufficient water supply.

Where electricity is used, one motor may provide power for the entire mill. In other places the mill is divided into separate units driven by independent motors. One Tennessee company has one motor for each set of three saw gangs. One advantage of such an arrangement is that one part of the mill may be shut down for repairs to the motor while the remainder is in operation. Some mill men object to the use of electricity if the power is conveyed over transmission lines. To guard against burning out motors by overcharge due to lightning, a circuit breaker is necessary, and during a thunderstorm the circuit may be broken and all the machinery stopped. When saw gangs stop suddenly, sand packs in around the blades, and may make starting difficult or impossible.

METHOD OF POWER TRANSMISSION.

For transmission from fly wheel to countershaft pulley two types of belts are employed—a broad belt of leather or fabric and a rope belt. The latter has the advantage of low first cost and is easily tightened, the tightening pulley being applied to a single turn only of the rope.

Direct water power is commonly transmitted by gears. Wooden cogs in one of the intermeshing wheels and iron cogs in the other have been found a very satisfactory combination. A skillful carpenter can easily replace defective wooden cogs, and such cogs, if properly made, are durable.

PLAN OF MILL, SHOP, AND YARD.

Stone is heavy and excessive handling adds greatly to the cost of the finished product. Consequently, marble producers should aim to arrange mill, shops, and yard in a manner that permits minimum handling. Where sawing and finishing are both done, it is a common practice to place mill and shop parallel, 30 to 60 feet apart, and to have an overhead traveling crane between them. Where sawing and

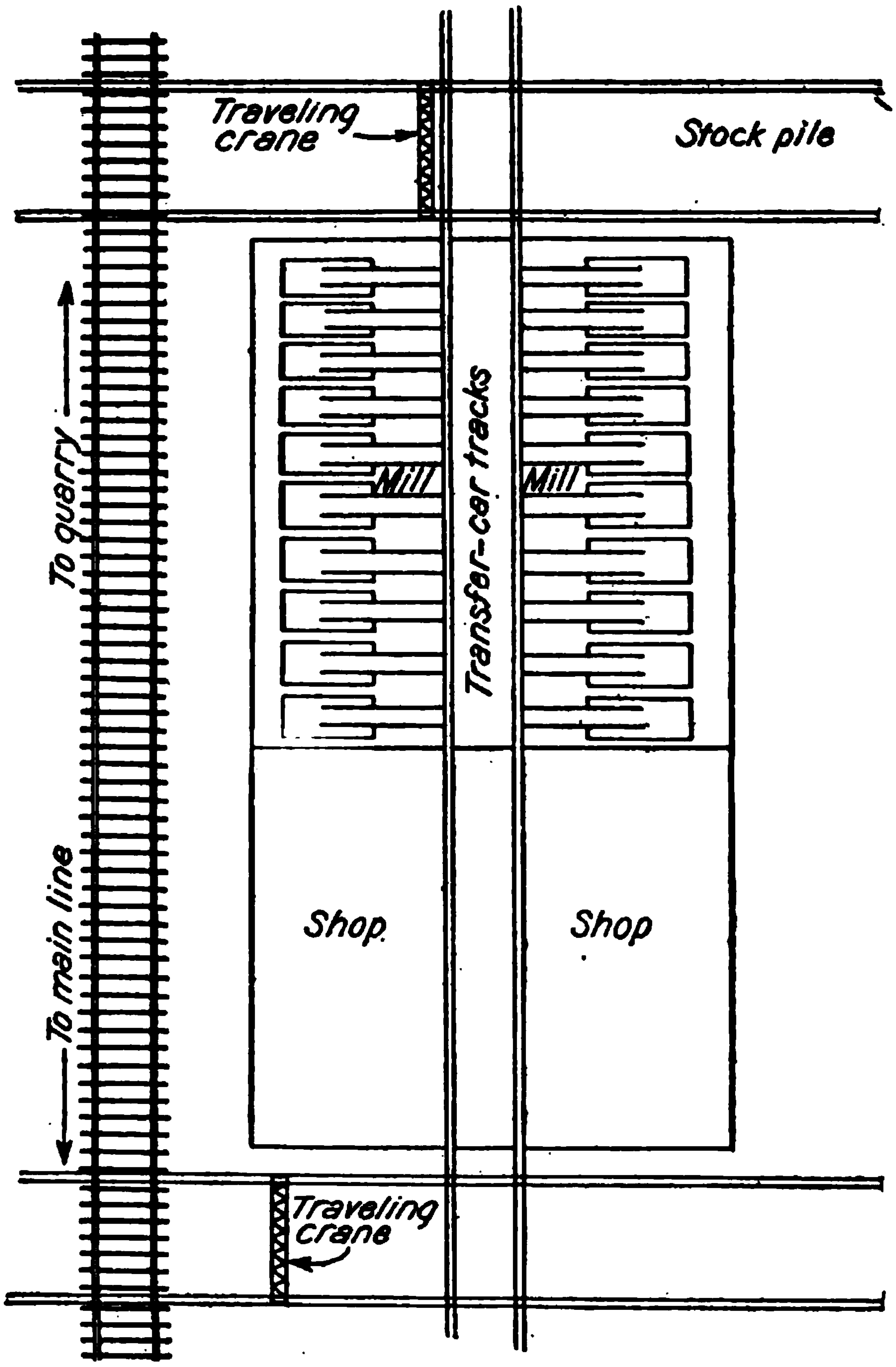

FIGURE 18.—Track arrangement at a Vermont marble mill.

finishing are conducted on a large scale, the plan of a Vermont mill is efficient. The essentials of the plan are shown in figure 18. One traveling crane is employed to unload blocks from cars on their arrival at the mill and to pile them with the stock or to load them on to the transfer cars. A small locomotive crane is employed to haul the transfer cars through the mill, where a track passes down the center with gangs on either side. Beyond the mill is the finishing

plant and at the end of it another smaller traveling crane for loading the finished stock on railroad cars.

A convenient plan for a small plant having a mill but no shop is that of a Maryland plant, as shown in figure 19. Gang cars are used but no transfer cars. Cars are loaded by either of two derricks situated at opposite sides of the mill. Railway tracks pass at both sides of the mill for bringing marble blocks, sand, or fuel, and for taking away finished stock. Both rough and finished stock are piled within reach of one or other of the derricks.

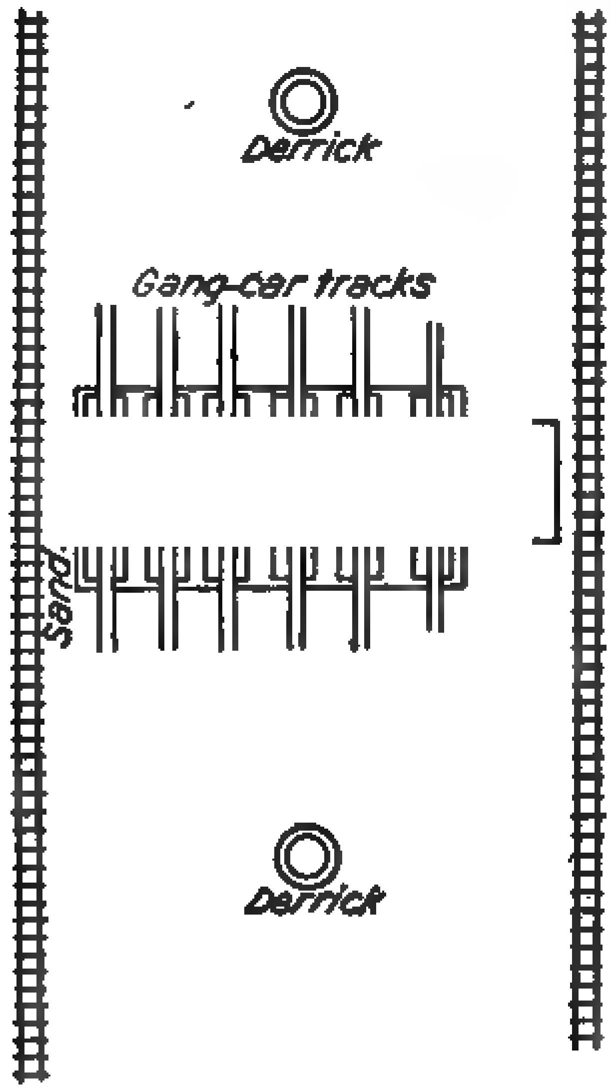

FIGURE 19.—Plan of mill and yard of a Maryland marble company.

MILL.

SAW GANGS.

Saw gangs constructed a number of years ago have wooden frames. Such frames are not to be recommended, as frequent wetting and drying cause them to decay rapidly. Modern gangs have steel frames which are more rigid and durable. An efficient frame devised by a Georgia company consists of iron tubes 8 or 10 inches in diameter, which were set up and filled with concrete. Such frames are constructed cheaply.

All hanger bearings for the gangs should be so capped that they are perfectly sand proof.

ABRASIVES USED IN SAWING.

The most common abrasive used in sawing is clean sharp sand. Many millmen use crushed steel or fine steel shot, either with water alone or with sand and water. The resulting increase in the rate of sawing is variously estimated by different operators at 20 to 50 per cent. As all millmen know, if the saws are fed too fast the blades bend and cut uneven surfaces, which must later be rubbed for a long time to be made uniform. Although it is possible to crowd saws to excess when crushed steel is used as an abrasive, the liability to crowding is less than when sand is used alone.

Certain rules must be followed in the use of crushed steel as an abrasive. It should never be used with rock that contains pores large

enough to allow grains of steel to become lodged therein. The same is true of marbles containing soft minerals, as mica. If the grains of steel become lodged in the pores or in the soft minerals, they may later cause scratches when the rock is under the buffer. Also porous marbles, especially those that exhibit a marked permeability, are apt to stain with the iron rust formed from the steel abrasive. A steel abrasive may be used successfully with marbles having small pores and no streaks or bands of relatively soft minerals.

It is well known that the presence of carbon dioxide greatly increases the tendency of iron to rust, and its complete absence renders the process of oxidation (rusting) extremely slow. On this account many millmen mix lime with the steel abrasive, because lime absorbs carbon dioxide to form lime carbonate according to the chemical reaction $CaO + CO_2 = CaCO_3$. When the carbon dioxide has been thus removed, the tendency for stains to occur in the marble is greatly diminished.

SAND PUMPS.

One good sand pump will supply five gangs. In some of the more improved types of sand pumps a great saving of power is effected by the use of ball bearings.

SAWING WITH UNSOUNDNESS.

If the grain of the marble permits, a great saving of material may be effected by sawing parallel with any joints present in the blocks. This can not be done if joints pass diagonally through blocks, and great waste results. Usually joints strike across the grain, and slabs must be sawed parallel with the grain. Thus joints may intersect a majority of the slabs, and sawing with unsoundness may not be permissible. As a rule unsound blocks can be sawed to better advantage into cubic stock than into thin slabs.

SAWING CORNER BLOCKS.

The ordinary method of sawing a corner block is to saw diagonally, forming triangular blocks. This method wastes material and necessitates a fourth cut. An improved method is illustrated in figure 20. Saw *1* is disconnected when it reaches *x* and saw *2* is disconnected at *y*. When cuts are made in a direction at right angles, L-shaped blocks are formed as shown.

LOADING SAW BEDS.

In early days the saw beds consisted of transverse stationary timbers. Roadways passed between the gangs over which marble blocks were hauled by teams or cables to points near the beds.

The blocks were placed in proper position by means of crowbars. Although greatly improved methods are now used, many of the old-fashioned mills are still in active operation, chiefly because the cost of remodeling is deemed to be too great. With the present high cost of labor, and of maintaining horses, the economy of adhering to the old methods is doubtful. Although the first cost of refitting would undoubtedly be high, there would be a subsequent saving of much time and labor. So many modern mills are now in operation that it would be a simple matter for any mill owner to estimate the loss or gain attendant on remodeling his plant.

THE GANG CAR.

The gang car has of late years replaced the stationary bed. The floor of the car constitutes the bed. Blocks may be loaded on the car and the loaded car placed beneath the blades and securely braced. The loading may be done by means of a derrick, a locomotive crane, or an overhead traveling crane. This method permits the loading of blocks and the removal of sawn material with great facility.

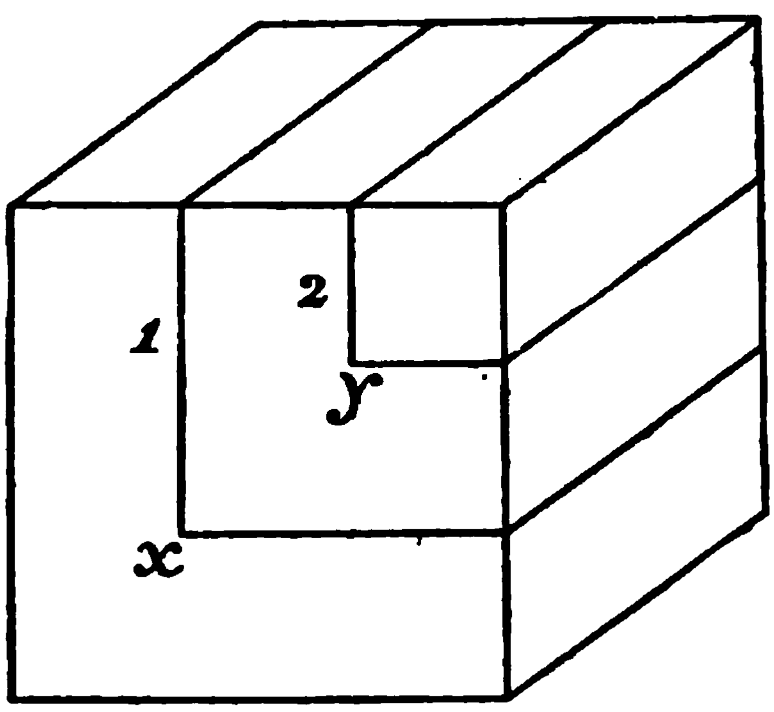

FIGURE 20.—Improved method employed by one marble company for cutting corner blocks.

THE TRANSFER CAR.

The use of a "transfer car" increases the facility of movement, and is a great timesaver especially where a traveling crane is not used. For small mills a convenient arrangement is to place a row of gangs on one side of the transfer-car tracks and a platform on the other. On the transfer car are cross tracks for the gang car. These tracks are level with the gang-car tracks which run beneath the saws on the one side and also level with several similar tracks on the platform at the other side. When a block is sawed, the gang car is hauled out by means of a cable and placed on the transfer car, from which it may be transferred to a position on the platform or taken to the shop. Another gang car with a fresh block may then be placed beneath the saws.

The platform is a great convenience when night shifts are employed. Gang cars loaded and ready are placed on the platform. When the cutting of a block is finished during the night it is removed and a new one placed beneath the saws in a short time. If no such conveniences are provided a gang that reaches the bottom of a block during the night must remain idle until morning.

The transfer car is not used at some plants, especially around mills where an overhead crane is employed. The gang must then remain idle during the time the gang car is being unloaded of slabs and reloaded with a new block. Where a transfer-car system is employed, the car loaded with slabs may be shifted to one side and replaced by another car already loaded and waiting.

The wisdom of using both transfer cars and a crane depends on circumstances. With extremely hard marbles, in which the saws sink only 3 or 4 inches during a shift, the time of loading and unloading is proportionally such a small part of the whole time devoted to the sawing operation that any equipment designed to shorten this time may not be justified. On the other hand, with soft marbles in which the saws advance 1 to 2 inches an hour a slow method of removing sawed material and replacing it with rough blocks may result in a considerable proportion of the saws being idle all the time. Mills have been observed where one-fourth to one-third of the gangs were kept permanently in idleness on account of the time required for unloading and reloading. In another mill equipped with both traveling crane and transfer cars, of the 20 gangs, 19 were in active operation at the time the mill was visited. The transfer car is undoubtedly a timesaver, and is not expensive in first cost, operation, or maintenance.

Some mills are so conveniently situated that the gang cars are loaded directly by the quarry derrick.

AN EXAMPLE OF A WELL-EQUIPPED PLANT.

The mill shown in figure 18 is an example of a mill suitably arranged and equipped for rapid handling of material. This mill has 43 gangs, and hence it is imperative to have ready facility in exchanging gang cars. The depressed transfer-car track passes down the center of the mill between two rows of saw gangs. When a block has been sawed, a small locomotive crane enters the mill from the yard. It pushes ahead of it an empty transfer car and hauls behind it another transfer car supporting a gang car loaded with a new block. When the empty transfer car reaches the proper position, the car with sawed material is hauled from beneath the saws by means of a cable from the crane and placed on the waiting transfer car. The locomotive crane then moves along until the gang car loaded with a fresh block is in proper position, when it in turn is hauled beneath the saws. The crane then passes on through the mill into the shop, where the sawed material is disposed of.

UNLOADING SAW BEDS.

Cubic stock may be removed by derrick or crane by using grab hooks or smooth-faced iron clamps which automatically close upon the block when under tension. Thin stock may be removed in the

same way or by cable slings. The removal of thin slabs singly, especially if no transfer-car system is employed, results in great loss of time and keeps saw gangs idle for too great a part of the time. If a pair of steel cable slings is used the whole load of slabs may be removed at a single operation. Some companies use the slings also for cubic stock.

A CONVENIENT DEVICE ON CABLE SLINGS.

The kinking and twisting of steel-cable slings causes annoyance, loss of time, and, occasionally, injury to the operator. A simple device used by one company overcomes this difficulty. A round iron ball weight is attached to the bottom of the sling. This holds the sling under sufficient tension to prevent twisting or kinking. The ball also affords a convenient means of handling the cable. It is less destructive to gloves and less liable to cause injury to hands than the cable itself.

THE WIRE SAW.

A useful adjunct to the mill is a yard equipment for cutting with a wire saw. Four to 12 blocks may be placed in line and sawed simultaneously, sand or crushed steel being employed as abrasives. The operation requires little power or attention and gives satisfactory results in uniform material if slight variation in the thickness of the slabs may be allowed. The wire saw will not give as satisfactory service with marble that contains flint balls or otherwise lacks uniform hardness as with pure and uniform material.

SHOP OR FINISHING PLANT.

LOCATION OF SHOP.

A company which operates both a mill and a shop should logically have the latter so situated that sawed material may be brought to it with the greatest facility. The mill shown in figure 18 illustrates a convenient arrangement, the shop being a continuation of the mill building. A parallel arrangement of mill and shop with a traveling crane between them is convenient, and many companies have their mills and finishing plants arranged in this manner.

COPING.

Although most coping is done by hand, a coping machine is employed in a few places. It consists of a small carborundum or carborundum-faced cutting wheel and a smoothly traveling bed on which the slabs are placed. It cuts rapidly and leaves a smooth surface. It may be used for cutting baseboards and tile. It gives good satisfaction if sound stock is employed. One advantage is that no edge rubbing is necessary after cutting with the machine.

RUBBING.

The two important requisites for a rubbing bed are hardness and uniformity. A good quality of iron has been found most satisfactory. The attempt to make carborundum beds of large size has not as yet been successful, owing to the limited size of carborundum furnaces. Composite carborundum beds could be constructed by joining together a number of segments. However, there would be difficulty in obtaining exactly the same degree of resistance to abrasion in each section. Small carborundum beds are used to some extent for rubbing small pieces.

The common abrasive on rubbing beds is sand. Unless the sand is nearly pure and uniform the use of a simple rotary screen to take out the pebbles is advisable.

The bed surface is kept true by grinding down the high parts with an iron weight. A cubical block of iron resting on the surface of the revolving bed serves this purpose. An improved form of truer consists of cylindrical rotating disks which are adjustable on the frame.

Most rubbing beds are driven from a countershaft and gears above the bed. When driven in this manner the shaft support both above and below the bed keeps the latter in a true horizontal plane. Occasionally, however, beds are geared underneath. It is more difficult to keep the bed running true when driven in this manner, as there is no support above the bed. However, the absence of a shaft above the bed allows great freedom in using a jib crane in handling blocks of marble.

Various methods of rubbing tile are employed to make them true to size and exactly square. Machines for holding them and automatically grinding them true are known to most millmen, and descriptions may be obtained from the various manufacturing companies. One method is to attach 8 to 12 tile together with plaster of Paris and then to rub to size in the same manner that a cubical block is treated.

In order to rub the surfaces of blocks too large to be placed on the rubbing bed, carborundum rubbing heads may be operated in the same way as buffer heads. The carborundum plates may be attached to the head with shellac, melted sulphur, or set screws. A star-shaped head with the water supply entering at the center is a new form that gives good satisfaction.

For curved and irregular surfaces hand rubbing is necessary. A piece of marble with sand and water or a carborundum brick is usually employed.

GRITTING AND BUFFING.

When a polished surface is desired, the marble is placed beneath a buffer to which is attached a head of felt or other soft texture. Emery powder is used for gritting, and the so-called "putty powder"

for polishing. The polishing powder is composed of chromium oxide, which makes a green powder, or of tin oxide, forming a white powder, together with oxalic acid. Occasionally these powders are mixed.

Different marbles act differently under the buffer. It is only by experience that one can learn the best method of polishing, and the peculiarity of the marble may demand a modification of the polishing machinery. As an illustration, a certain Vermont marble polishes best when nearly dry. The ordinary buffer when nearly dry has a tendency to jump and break thin stock. To overcome this difficulty, "pendulum buffers" have been devised. By means of a crank and pitman large wooden blocks are made to slide back and forth. These blocks are covered with felt pads, by means of which the polishing is done, putty powder almost dry being used.

For hand polishing of curved or irregular surfaces a fine sandstone or hone is employed.

CUBIC STOCK.

Unsound blocks cut to better advantage into cubic than into thin stock. On the other hand acute-angled blocks should be cut into thin stock to avoid waste.

For cutting cubic stock to proper dimensions, a perforated steel circular saw, a diamond circular saw, or a single blade in a straight-cut gang frame may be employed. With the perforated steel circular saw sand or steel shot is employed as abrasive. It cuts fairly well but is now replaced in many places by the more rapidly cutting diamond saw.

Circular diamond saws vary in diameter from 20 to 72 inches. The diamond teeth are replaceable. The first cost is high, though with care the cost of maintenance is not excessive. They occupy little space and saw rapidly. An abundance of water is necessary for successful operation, and care must be exercised to avoid overcrowding. A New York company operates a pair of parallel diamond saws which are adjustable for width.

A single saw blade with crushed steel or sand as abrasive is occasionally used. Its operation is too slow to be satisfactory. A blade set with diamond teeth and placed in a straight-cut gang frame may be employed.

PLANERS.

Planers are used extensively for cutting moldings and cornices. The more improved forms may be applied to either straight or circular work. Of late years a great deal of the work formerly done with planers is performed with carborundum machines.

CARBORUNDUM MACHINES.

Carborundum machines are great time savers. The extensive use of carborundum is one of the most remarkable modifications in the equipment of modern marble-finishing shops. Curved work, moldings, cornices, or balusters are all cut successfully with carborundum wheels.

The method of preparing a carborundum wheel is to set it in a lathe, and by means of a steel tool to cut it into the shape of a negative of the pattern desired. The wheel is then placed on the shaft. The marble block travels on the machine bed beneath the wheel which cuts the molding to the desired shape. A copious stream of water plays on the cutting surface. Plate IX[a] shows the manner in which a molding is cut. Round bases may be carved by rotation of the marble mass, and curved forms may be made by a continuous automatic adjustment of the cutting wheel, as shown in Plate X, *A*. A carborundum wheel in use for making balusters is shown in Plate X, *B*. The piece of marble is roughed out to the general shape desired. The carborundum wheel and the baluster are rotated in opposite directions. Balusters are manufactured much more rapidly by this method than with a lathe, especially if the marble is hard or tough. One company manufacturing balusters from the Roxbury, Vt., verd antique completes in about one hour a baluster 3½ feet long and about 6 inches in diameter at its widest part. The time required to turn a similar baluster on a lathe is about three hours.

Machines are now manufactured for fluting large columns. Rather than cut out all the material with a carborundum wheel, one company finds it advantageous to first use diamond saws to make a series of parallel cuts in the deepest part of the fluting. The intervening marble is broken out with a hammer and the operation completed with the carborundum wheel. By thus using diamond saw and hammer to remove the bulk of the material the time required for fluting is just one-half that required if all the cutting is done with the carborundum wheel.

Another company has introduced a simple machine to save time in cutting circular or other irregular work. A horizontal iron bed about 5 feet in circumference is surrounded by a galvanized-iron pan to catch the drip. Projecting from the center of the bed is a vertical shaft to which a carborundum wheel is attached. The marble block is roughed out and the desired outline carefully marked. In order to facilitate movement of the marble block on the bed, flat iron plates 12 by 1½ inches by ¼ inch are provided with ball bearings inserted in

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[a] Pl. IX and Pl. X, *A* and *B*, were kindly supplied by the Julius Wegner Machine Works, Astoria, Long Island, N. Y.

holes and projecting slightly above the surface. The arrangement of ball bearings is shown in figure 21. The block is placed on these plates and is thus very easily guided. It is held against the rotating carborundum wheel and cut to the line. The claim is made that one man can accomplish as much with this machine as 10 men cutting by hand.

One weakness of the method is the lumpy or uneven surface that inevitably results from hand manipulation. The company contemplates the introduction of an adjustable block support to be attached to a lathe bed running beneath a carborundum wheel. The bed is designed to travel very slowly while the turning of a handwheel raises or lowers the block. Thus the accuracy of machine work will replace the unavoidable inaccuracy of hand labor. Similarly, the introduction into marble-finishing plants of many new and special forms of carborundum machines may be expected.

COLUMN CUTTING.

Various methods are employed for cutting columns of marble. One company cuts small columns by means of a circular steel drum rotating on a vertical axis and fed with sand and water. Other companies employ similar drums having diamond teeth inserted on the lower margin. The cutting is done by the diamonds and no other abrasive is necessary. A stream of water is supplied during the cutting process.

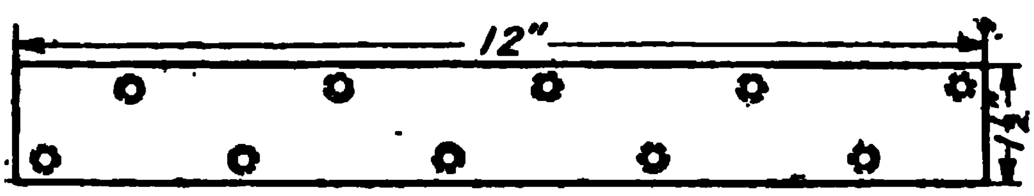

FIGURE 21.—Iron plate with ball bearings used to facilitate movement of marble blocks.

The largest cutting drum of this type yet observed is that employed by a Colorado company in cutting columns for the Lincoln Memorial, now under construction at Washington, D. C. Each section of a column is 7 feet 5 inches in diameter and 58 inches long. The drum has 80 diamond teeth. A period of 4 to 5 hours is required for cutting each section.

Drum column cutters give good satisfaction for short columns or short sections of columns, as described above. For large monoliths a lathe must be employed. Columns are then roughed out to the approximate form desired before they are placed in the lathe. The cutting tool employed is similar to that used in the ordinary machine lathe for turning metal shafts.

Lathes adapted for fluting as well as turning are now used by some companies.

A lathe may also be employed for polishing plain columns, though the rubbing or polishing of fluted columns is done by hand.

CARBORUNDUM MACHINE CUTTING A MOLDING IN MARBLE.

A. A CARBORUNDUM MACHINE ADAPTED FOR CUTTING CURVED MOLDINGS.

B. A CARBORUNDUM MACHINE ADAPTED FOR CUTTING BALUSTERS.

CUTTING AND CARVING.

All complicated patterns, or other irregular designs, must be cut by hand. Much of the cornice and molding work formerly done by hand is now performed successfully with planers or carborundum machines. As mentioned previously, carborundum machines are of especial interest to stonecutters, as they are capable of such varied adaptations. For hand carving, small operators with limited capital employ hand tools and hammers. Pneumatic tools are much better, and should be employed wherever possible.

A method of cutting the letters on headstones by means of a sand blast is employed by at least two companies. A shield with an opening the size and shape of the inscription area is placed over the monument. Steel letters are glued on the rock surface in proper position, and a sand blast directed at high pressure against this surface for a few moments cuts down the entire area except that protected by the steel. A little hand trimming is required to correct irregularities due to varying hardness of the stone. Much time is saved by employing this method, especially when many monuments of the same size and shape are manufactured.

That part of a stone shop that is devoted to cutting and carving should be well lighted and heated.

HANDLING MATERIAL.

An overhead crane is necessary for handling heavy material Electric cranes are efficient, and are so widely known that description of their operation is superfluous. In many shops where small-sized stock is produced the material is handled with great facility by means of small trucks run by hand.

CRATING AND SHIPPING.

Experienced men should be employed for crating and loading finished material upon cars ready for transportation. The material must be packed in such a manner that it will not break by moving about in the car, but, on the other hand, it must not be secured too rigidly, or the straining of the cars on sharp curves may cause breakage. For handling heavy material a derrick, locomotive crane, or overhead traveling crane must be employed. For small stock, trucks are commonly used.

ADEQUATE WATER SUPPLY.

So many operations in the mill and finishing plant require an abundance of water that provision must be made for an adequate supply. Failure of the water supply may make necessary suspension of the operation of part or all of the marble-cutting machinery. If for any reason the supply of water seems liable to diminish and become

inadequate, immediate steps should be taken to obtain a larger supply or to build necessary reservoirs for the regulation and conservation of the supply already at hand.

THE PROBLEM OF WASTE.

THE IMPORTANCE OF WASTE.

Conservation of national resources demands economy in quarrying. The stone resources of the United States, although great, are by no means inexhaustible, especially the finer grades of marble. Added to the actual value of the material forming the waste heap is the cost of excavation. The cost of stripping, channeling, drilling, and hoisting waste blocks is often nearly as great as the cost of handling an equal amount of good rock. Aside from the value of the rock and the additional cost of quarrying, the waste material encumbers the ground, impedes lateral development, and interferes with yard operations. The failure of some quarry companies to realize a profit is due to the quarrying of an excessive amount of material that remains unutilized.

The problem of waste is twofold. In the first place it has to do with all types of improved equipment and modern methods of excavation which tend to keep the proportion of waste at a minimum; and in the second place it must deal with the various uses to which waste material may be applied. In other words, it is a problem, first, of waste elimination and, second, of the utilization of whatever waste is unavoidable.

ELIMINATION VASTLY MORE IMPORTANT THAN UTILIZATION.

Waste elimination is much more desirable than waste utilization. A quarryman should by no means countenance methods that result in excessive waste merely because he has found an outlet for his waste material in the form of by-products. As a rule the cash return from by-products is only a fraction of the production cost of the waste material from which they are supplied. As an illustration, it may be assumed that a moderate cost of marble excavation is 25 cents per cubic foot, or $3 per ton. A fair price for riprap is 50 cents a ton, one-sixth of the cost of excavation. The quarryman seeks a market for riprap, not because the production of riprap is profitable, but for the reason that he prefers to obtain one-sixth of the cost of his waste material rather than to receive nothing at all. By eliminating a ton of waste he saves $3, whereas by marketing it he saves only 50 cents.

WASTE ELIMINATION.

UNAVOIDABLE LOSSES.

The loss of a part of the good stone is unavoidable. Channeling, drilling, scabbling, sawing, and coping are all necessary operations which use up an appreciable share of the stone. In addition to losses

due to the processes of manufacture, more or less stone must be thrown away on account of imperfections.

AVOIDABLE LOSSES.

It is, however, the throwing away of masses containing many cubic feet of good stone, or the handling of an excessive amount of inferior material, which constitutes the serious and, for the most part, avoidable losses. Causes of these greater losses and methods of quarrying that will eliminate them are discussed below.

CHIEF CAUSES OF WASTE.

The natural imperfections in marble that constitute the source of the greater losses are unsoundness, strain breaks, impurities, and lack of uniformity either in color or texture. On account of the particular method followed in marble quarrying, irregular masses or acute-angled blocks may result and lead to further waste.

SYSTEMATIC PROSPECTING.

Unsoundness, texture, and distribution of color or impurities vary from point to point in the same deposit. The prospective quarry operator must not too hastily open a field of operations. Systematic prospecting is a first step toward waste elimination. Outcrops or stripped surfaces should be carefully examined for unsoundness, bad color, flint balls, etc. Naturally, investigation should follow the direction in which any improvement appears in the rock.

The most suitable location as indicated by surface conditions having been chosen, the next step is to ascertain the qualities of the marble at various depths. Drill holes should be projected at points distributed systematically over the area under investigation. Double core-barrel drills are the best for this purpose, as they give information concerning unsoundness in addition to indicating color, uniformity, and supply. Any change in quality with depth is important as showing whether development should best extend laterally or vertically.

DIRECTION OF QUARRY WALLS.

Before operations are started the outcrop or stripped surface should be mapped carefully to show the direction of strike and dip, and the directions of the chief joint systems. Naturally the quarry walls should parallel those rock structures that are most pronounced. If the beds are tilted and if inferior beds alternate with those of good quality, it may seem advisable to make the quarry walls parallel the strike and dip. If the rock is of uniform quality but intersected by prominent joint systems, the quarry walls should be parallel and at

right angles to the chief joints, or possibly the contiguous walls should parallel the two chief systems of joints if these should meet at oblique angles. The quarryman must use keen judgment and give careful study to this question. Careful mathematical calculations may be necessary before he can determine definitely which plan will give him the minimum of waste.

The author knows of instances where a mistake in the original plan of quarrying has led to an excessive proportion of waste. In a certain quarry the marble beds strike N. 35° W., and dip 30°. The sides of the excavation are N. 60° W. and N. 30° E., respectively. Hence, one wall of the excavation makes an angle of 25° with the strike. As a consequence the beds pitch into one corner of the quarry, as shown in Plate VII, *C*. The nature of the rock requires that it be split on the bed, and as a consequence "oblique" blocks result and the waste is excessive.

Of course when such a mistake has been made in the original plan of quarrying, it is possible to change the plan and quarry parallel with the chief rock structures. By such a change, however, corners are left and the original floor space greatly reduced.

TUNNELING TO AVOID UPPER INFERIOR BEDS.

The depth of inferior rock due to surface alteration is an important consideration. Although the actual value of the untouched material may be negligible, the cost of handling great quantities of waste material adds greatly to the expense of quarrying. The removal of such material may, under certain conditions, be avoided by employing tunnel methods.

WASTE IN TUNNELING.

In projecting a tunnel, head room is obtained by blasting out a space 6 to 7 feet in height. If the tunnel is driven in good marble a large quantity of good material is thus destroyed. If practicable, the quarryman should choose an inferior bed in which to drive the preliminary opening. If a tunnel is to be projected with a view to avoiding handling surface-stained material, it should be driven in the band immediately above the good rock, provided a sound roof of sufficient thickness remains.

The blasting required in tunnel work demands care to avoid shattering the good marble. When tunneling is done on the bed, a band of mica schist or even an open bedding plane at the floor of the preliminary tunnel may serve as a cushion and prevent the shock from affecting the good marble beneath. If no such cushion exists, as a safety precaution, only light charges are used, and holes are fired singly as the simultaneous discharge of a series of shots creates too

severe a shock. The making of a horizontal floor cut with a channeling machine prevents the effects of blasting from reaching the good marble, but the difficulties and inconveniences of the method make it of doubtful value.

WASTE DUE TO UNSOUNDNESS.

Channeling regardless of unsoundness probably accounts for the loss of a greater quantity of good marble than any other single cause. Unsoundness is the most prolific source of waste, and the one that is receiving least attention in the majority of American marble quarries. Too great emphasis, therefore, can not be placed on this phase of the waste problem.

Waste results whenever joints pass through blocks, and the waste becomes excessive when they pass in diagonal directions. A reduction to a minimum of this form of waste involves first a modification of channeling and drilling directions in order that they may conform with the directions of the chief joint systems, and second a variation in the spacing of cuts to make them coincide with joints and thus eliminate the joints from the blocks. Attention is directed to figure 10 (p. 63) which illustrates the manner in which waste may be eliminated by a judicious arrangement of channel cuts.

WASTE DUE TO LACK OF UNIFORMITY.

If there is a variation in the texture or color or the marble, care should be taken to quarry in such a manner as will tend to produce material that may be closely classified. Thus, channel cuts and drill holes should follow as closely as possible the boundaries between different grades of material. If a block of stone contains 75 per cent of inferior and 25 per cent of good material it will probably be thrown in the waste heap, along with blocks that are 100 per cent inferior. If, however, it is so quarried that even part of the 25 per cent of good material is retained on a good block, a saving has been effected. This feature is discussed on page 60. By making all the cross breaks along lines that mark separations between material of different grades a close classification may be made.

WASTE DUE TO IRREGULAR BLOCKS.

Imperfect quarry methods, such as making too few drill holes or wedging so forcibly as to produce artificial strain breaks, may result in the production of irregular blocks. Channeling or otherwise separating blocks along lines making acute angles with open seams or beds may result in the production of irregular or angular fragments, most of which are thrown in the waste heap. When irregular blocks are placed on the saw bed the removal of thick surface slabs

and of irregular or angular ends leaves a relatively small proportion of good stone.

THE VARIOUS REGULAR FORMS OF QUARRIED BLOCKS.

Usually marble is quarried in regular blocks bounded by three pairs of parallel faces. Such regular blocks may be in various forms. The relative waste involved in each form, and the conditions that justify or condemn excavation in such a way as to produce each form are of supreme importance.

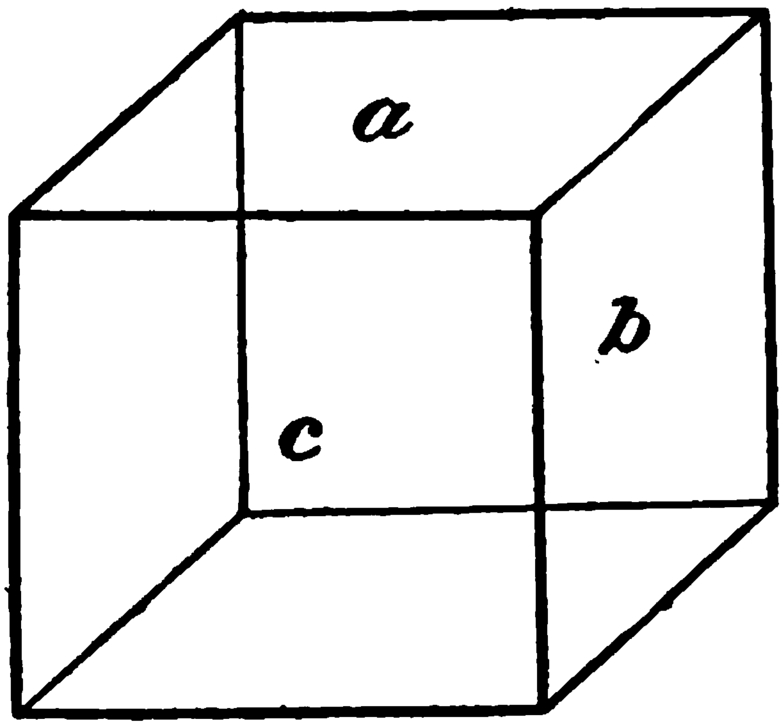

FIGURE 22.—Right-angled block.

Of regular blocks bounded by three pairs of opposite and parallel faces, four different forms are quarried. These are represented in figures 22, 23, 24, and 25. The block represented by figure 22 is bounded by three pairs of parallel faces, all adjacent faces meeting at right angles. Hence, this may be termed a "right-angled" block. In figures 23, 24, and 25 a right-angled block is shown in dotted lines, in order to indicate the relationship existing between it and the other forms described.

Figure 23 represents a block that results when the quarry floor is level, when one channel cut crosses the line of strike at right angles, and when the cross break or second channel cut is made parallel with a steep dip. The angles between faces *a* and *b*, and between faces *c* and *b*, are right angles, whereas between faces *a* and *c* the angle is greater than 90°. The face *c* and its parallel and opposite face are inclined at an oblique angle with respect to one other pair of faces, and in consequence the block is termed a "monoclinic" block.

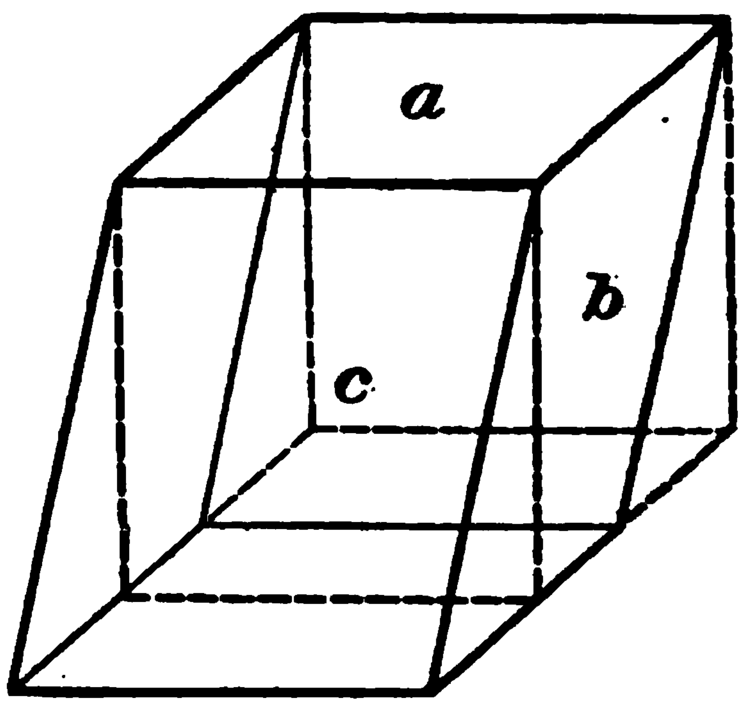

FIGURE 23.—Monoclinic block.

Figure 24 represents the form of block which results when the quarry floor *a* is level, the channel cut forming the face *b* is vertical, and the cross break or channel cut forming the face *c* passes obliquely through the block with respect to both *a* and *b*. Between the faces *a* and *b* the angle is right, whereas the angles between *a* and *c* and

between *b* and *c* are both oblique. The line marking the emergence of the plane *c* on the level quarry floor is not perpendicular to the channel cut that forms the face *b*, as with the block shown in figure 23, but passes obliquely across the surface. Hence, this block may be termed "oblique."

Figure 25 shows a block in which no right angles exist between the bounding faces. Such a form results when the quarry floor is level and when the channel cuts are neither at right angles to each other nor vertical, being inclined at steep angles. As there are no right angles and as the three pairs of faces are inclined at oblique angles to each other, such a block may be termed "triclinic."

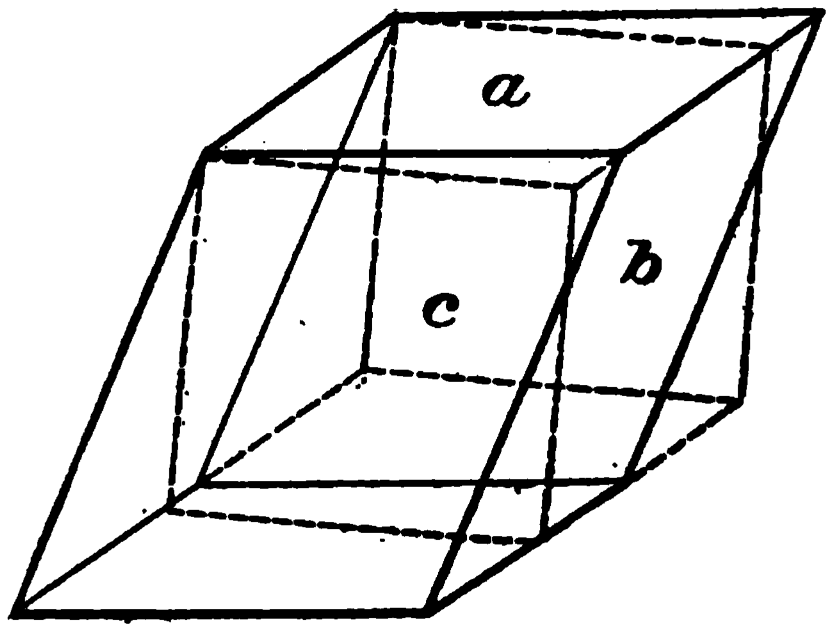

FIGURE 24.—Oblique block.

It may be noted that the four forms described represent the varying stages of obliquity that are possible in regular forms bounded by three pairs of parallel faces. Of the angles formed by the meeting of the three faces *a*, *b*, and *c* in figure 22, there are three right angles; in figure 23, two right angles and one oblique angle; in figure 24, one right angle and two oblique angles; and in figure 25, all three are oblique.

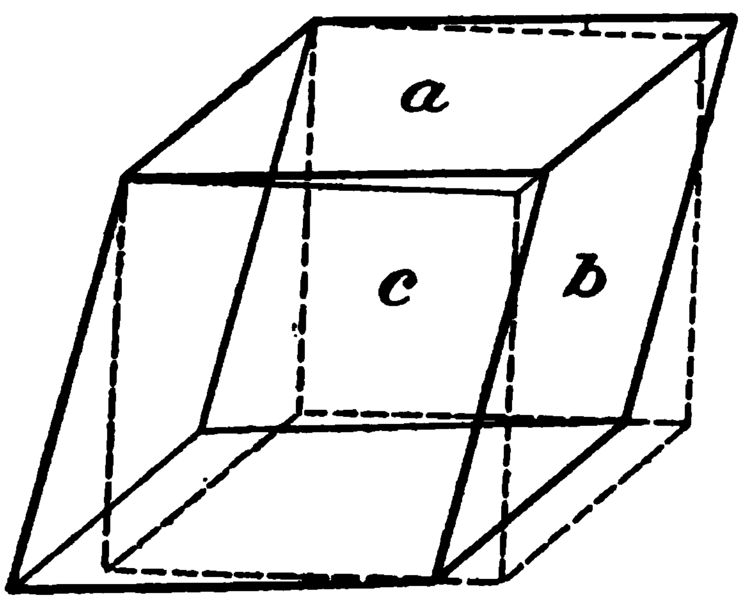

FIGURE 25.—Triclinic block.

CIRCUMSTANCES GOVERNING THE PRODUCTION OF THE VARIOUS FORMS.

For convenience in describing the manner in which blocks are formed, the three cuts by which blocks are separated may be termed the "floor cut," the "strike cut," and the "dip cut." The floor cut or "bottom break," as it is sometimes called, is the break that separates the block from the quarry floor, as shown at *a*, figure 26. The strike cut is parallel with the strike of the rock at shown at *b*, and the dip cut follows a line at right angles to the strike cut, as at *c*.

Right-angled blocks are obtained when the quarry floor is level and when the channel cuts and cross breaks are vertical and at right angles. They may also be produced if the quarry floor slants so as to parallel the dip, if the channel cuts and cross breaks are made in such a manner as to form right angles with each other and with the

floor, as shown in figure 26. In some quarries the dip cuts are made with a channeler and the cross breaks by drilling and wedging. If the rift is slight it may be desirable to make channel cuts in both directions. Inclined strike cuts may be made on a slanting floor with Sullivan "Z" or Ingersoll electric air machines. They operate on a level track with the channeling machinery inclined in such a manner as to make the strike cuts perpendicular to the dip as shown in Plate XI.[a]

If beds are inclined at an angle and the quarry floor is maintained on a level it may seem desirable to quarry acute-angled blocks. Inclined marble beds may have a pronounced rift, or a color distribution, or both, in a direction parallel with the bedding. Under such conditions, ease of splitting or a proper grading of material may require the strike cuts to be inclined to parallel the dip as shown in Plate XII, *A*. As a result, with a level quarry floor, and one set of channel cuts perpendicular to the strike, "monoclinic" blocks are produced. Such blocks are also formed if the quarry floor is inclined to parallel the beds and the strike cut is channeled vertically.

FIGURE 26.—Cuts by which blocks are separated from the solid mass. *a*, Floor cut, *b*, strike cut; *c*, dip cut.

As mentioned previously, a serious mistake is occasionally made by failing to make the quarry walls conform in direction with the strike of the rock. Under such conditions, inclined open beds will pitch into one corner of the quarry, as shown in Plate VII, *C* (p. 86). When a level floor is maintained, a separation parallel with the beds will pass obliquely with respect to the channel cuts, resulting in what have been termed "oblique" blocks, shown in figure 24. Oblique blocks may also result from a modification of the regular vertical and right-angled direction of channel cuts in order to parallel a system of joints.

"Triclinic" blocks are produced only under peculiar conditions. In one quarry previously mentioned the marble is quarried on a level floor, the strike cut, which runs N. 5° W., is inclined 60° from the horizontal to parallel the dip, and the second channel cut is made in the direction N. 82° E., and is inclined 61° from the horizontal to parallel a pronounced system of joints. Thus the three pairs of faces

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[a] Photograph supplied by the Sullivan Machinery Co.

A METHOD OF CHANNELING RIGHT-ANGLED BLOCKS ON AN INCLINED FLOOR. QUARRY AT AVONDALE, CHESTER COUNTY, PA.

A. METHOD OF QUARRYING MONOCLINIC BLOCKS.

B. VERTICAL DRILL HOLES PARTLY CLOSED BY ROCK EXPANSION.

of the resulting block meet each other at oblique angles to produce a "triclinic" form, as shown in figure 25.

WASTE DUE TO ACUTE-ANGLED BLOCKS.

As by far the greater proportion of all structural material is right-angled in form, right-angled blocks are the most desirable and give a minimum of waste material.

When a monoclinic block is cut into cubic stock, angular corners are cut off and thrown into the waste heap. The excessive waste resulting from such a process is indicated in figure 27. Quarrying on an inclined floor, thus producing right-angled blocks, would greatly diminish the waste.

That a greater proportion of waste must result from oblique blocks than from those of monoclinic form is shown in figure 28. If the oblique form results from a mistake in the original layout of the quarry, and if much cubic stock is desired, it is probable that a

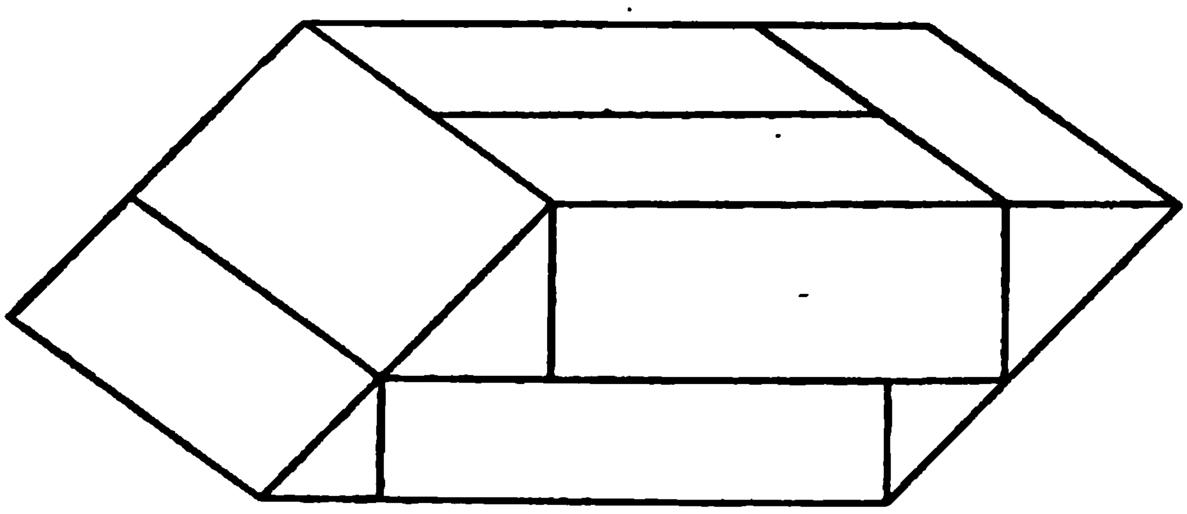

FIGURE 27.—Diagram showing waste resulting from cutting monoclinic blocks into cubic stock.

change in the direction of channeling to conform with the direction of strike would be justified.

No diagram is necessary to indicate that the triclinic block is even less economical than the oblique.

The angle of obliquity has a direct bearing on the economy of quarrying. The acute angles may approach 90° or they may be 60°, 45°, or even 30°. It is at once evident that the more acute the angle becomes the greater is the proportion of waste. The increase in waste due to increase in obliquity is especially pronounced when cubic stock is produced.

JUSTIFICATION FOR ACUTE-ANGLED BLOCKS.

It has been shown that with each increase in the number of oblique angles in a block of sound and uniform marble, provided the acute angles are of approximately the same size, there is a corresponding increase in the proportion of waste. The question arises, Is the production of acute-angled blocks ever justified?

The nature of the product has an important bearing on this question. When the quarried material is cut into thin stock exclusively, slabs of uniform size may be produced from monoclinic blocks with little waste, as shown in figure 29. Except for the beveled edges that must be removed in coping, the waste is not abnormal. However, it must be remembered that in sawing monoclinic blocks longer saws are required than for blocks of the same size in rectangular form. Also, if long blades are used, the cuts are liable to be irregular.

Under certain conditions acute-angled blocks are the most economical that can be quarried, especially if the material to be quarried is unsound or nonuniform. It is illogical to state that as acute-angled blocks do not as a rule cut to advantage their production should always be avoided. Usually economy demands that blocks be separated parallel with the prevailing rock structures, and these structures may not be in right-angled arrangement. They may be so arranged that by quarrying in conformity with them, monoclinic,

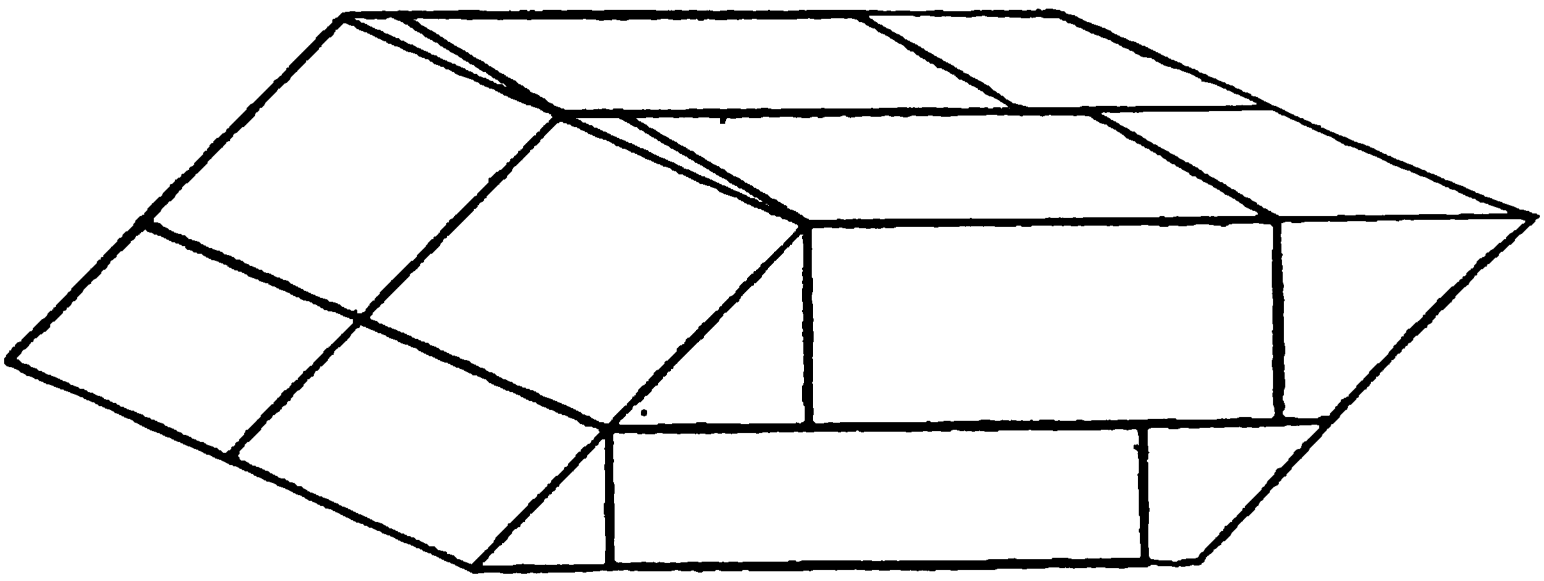

FIGURE 28.—Diagram showing waste resulting from cutting an oblique block into cubic stock.

oblique, or even triclinic forms must result. Rock structures are fixed and unavoidable features, and many of them are undesirable in nature, distribution, and direction, so that the quarryman must modify his methods to make the best of them. Although in quarrying sound marble right-angled forms are the most economical, in quarrying unsound or nonuniform material conformity with structure is of greater consequence than right angles.

A concrete example will illustrate this point. In one part of a certain quarry, color bands and joints make oblique angles with each other and with the level quarry floor. Quarrying parallel with these structures results in the formation of triclinic blocks. Inclining the floor to make a right angle with either the color bands or the joints would result in the formation of oblique rather than triclinic blocks with a consequent small saving of material. However, the conditions described prevail over a small part only of the quarry, and as a consequence a change of floor attitude would not be justified.

If in this instance the quarryman had channeled vertically in two directions at right angles, and thus produced right-angled blocks, the undesirable color bands would have passed through the blocks diagonally in one direction, and the joints would have passed diagonally in another direction. Such blocks would probably be worthless. In other words the right-angled block would give the maximum of waste and the triclinic block the minimum. Thus under certain conditions the quarrying of acute-angled blocks represents the highest type of efficiency.

GENERAL RULES GOVERNING SHAPE OF BLOCKS.

Rules for the quarryman's guidance may be summarized as follows:

(1) Effort should be made to produce right-angled blocks, unless there is a valid reason for doing otherwise.

(2) Quarrying on a level floor and splitting diagonally to form monoclinic blocks may be justified where much thin stock is produced. If much cubic stock is desired, the quarryman should consider carefully the advisability of channeling on an inclined floor in order to produce right-angled blocks.

FIGURE 29.—Diagram showing small amount of waste resulting from cutting monoclinic blocks into thin stock.

(3) A direction of channeling that results in inclined beds separated by open bedding seams pitching into the corner of a quarry should by all means be avoided. The same is true of inclined beds that are not separated by open seams but have a decided rift or color distribution parallel with the bedding.

(4) As regards unsound or nonuniform material, although an effort should be made to avoid oblique angles, conformity of cuts with structure is, as a rule, more economical than right-angled cuts.

IMPURITIES THAT CAUSE WASTE.

The more common impurities in marble are: Silica, dolomite, pyrite, or marcasite, and mica.

SILICA.

The presence of flint or some other form of silica in marble beds is a source of waste in many places. Flint balls or lenses of silicious matter that are seemingly conformable with the bedding are probably original constituents of the marble deposit. As mentioned in the section on marble imperfections, such occurrences are probably characteristic of all parts of the beds in which they appear. The

presence of flint balls in a marble bed should discourage any attempt at extensive excavation with a view to uncovering other parts of the same bed in the hope that flint may not be present. It is wiser to assume that such a bed is undesirable and to conduct quarry operations in such a manner as to handle as little of it as possible. If silica is a secondary filling in cracks and cavities, it tends to follow the joint system, and consequently if the rules given for economical quarrying in unsound marble are followed the silica may be largely eliminated with the unsoundness.

DOLOMITE.

Bands or lenses of dolomite are present in some marble deposits. Although dolomite is just as desirable a form of marble as calcite, a mixture of the two is undesirable. Dolomite is harder and less soluble than calcite in ordinary atmospheric reagents. On the other hand, owing to the tendency of dolomite to occur in granular form, it frequently dissolves more rapidly than calcite. On exposed surfaces differential weathering is likely to take place, and in time a pitted or otherwise nonuniform surface will result. Quarrying should be conducted in such a manner as to avoid those beds that show a mixture of calcite and dolomite unless the mixture is intimate.

PYRITE AND MARCASITE.

The effect of pyrite and marcasite on the quality of marble may be pronounced or it may be scarcely noticeable. Those forms of either impurity that have a tendency to cause stains by weathering should be avoided in quarrying marble for exterior work.

MICA.

The presence of mica in marble is probably the result of the effects of metamorphism on interbedded clay. As clay bands are formed from stream deposits, they are near-shore phenomena. The marble prospector should have a thorough knowledge of the geologic history of his deposit. It is desirable that he should know the position of the shore line when the original beds that formed the marble were deposited. Other factors being equal, it is desirable to locate quarries at some distance from old shore lines in order to avoid as much as possible of the interbedded mica.

WASTE DUE TO BAD COLOR.

A great saving may be effected by close grading of nonuniform material. Slight changes of color from white to gray or from white to bluish white may seriously detract from the market value of the stone. The reasons for such changes are obscure. They are probably due to the presence of chemical impurities in small amount.

If the impurities are original components of the marble and due to the peculiar conditions of limestone deposition, it follows, as in the case of flint, that the distribution of color will tend to be fairly constant in any one bed, and the greater variations will be found in passing from one bed to another.

WASTE DUE TO STRAIN BREAKS.

A condition of strain within the marble mass has in certain places caused so great a proportion of waste that the workings have been abandoned. The rock is under a severe compressive stress usually in one direction only. The process of quarrying relieves the stress at certain points, and the consequent expansion may cause fracturing. Furthermore the expansion of one mass that is still in rigid connection with the main mass still under compression may cause irregular or oblique fractures to form between the two masses.

In order to avoid the excessive waste due to this cause steps should be taken to bring about relief by uniform expansion of as large a mass as possible at one time. To this end a line of closely spaced deep drill holes should be projected along each side of the quarry parallel with the direction of compression, and a similar line of holes should be projected across the quarry at right angles to the first line. The rock will expand and close the holes in the latter line, and the strain will thereby gain relief. For a more complete discussion of the problem of strain breaks the reader is referred to pages 123–145.

WASTE UTILIZATION.

BY-PRODUCTS.

Although the proportion of waste may be kept at a minimum by the adoption of economical quarry methods and efficient machinery there is always more or less unavoidable waste. The second phase of the problem of rock waste has to do with the various means of utilizing the waste material. Many manufacturers in various lines of industry have found that the manufacture and sale of by-products from otherwise waste materials have placed their industries on a profitable basis. The tremendous heaps of waste material found near many marble quarries testify to the need of greater development along the line of waste utilization as well as waste avoidance.

The difficulties in the way of such development are of various kinds. The need of a market for rock products hinders all activity in some regions. Local conditions may be such that the demand is small, and freight rates may be excessive. Lack of a market in other places may be due to the fact that although certain rock products are useful, the uses are unknown, and the demand is small until a campaign of public education has created an interest in them.

The nonutilization of waste may be due to lack of equipment. Available capital may not be sufficient to build or purchase the necessary tracks, kilns, crushers, or pulverizers for the manufacture or transportation of the various rock products.

RIPRAP.

One important use for waste marble is in riprap for shore protection in rivers and harbors, for filling in piers, or for constructing railroad embankments. The utilization of marble blocks for such purposes depends chiefly on freight rates. The chief item in production is the breaking of massive blocks into fragments of convenient size. This may be done by pop shots for the larger and hand hammers for the smaller blocks.

THE BALL BREAKER.

The cost of breaking may be greatly reduced by adopting a method introduced by an Alabama marble company. A breaker consisting of a steel ball weighing about 2 tons is employed. The bale is set in a cavity to avoid breakage. The ball is hoisted to a considerable height with the derrick, then tripped and allowed to fall on the marble block. The impact is sufficient to break large blocks into fragments of convenient size for handling.

An iron sphere weighing about one ton was first employed. It was not strong enough to withstand the severe treatment, and was later replaced by a steel ball of double the weight.

The spherical form first used was unsatisfactory, because the ball would roll and the bale was frequently at the bottom and difficult to reach. A pear-shaped form with the bale at the small end as shown in Plate VI, *C*, is now used. This form tends to keep the bale always in an available position.

The ball can be handled quickly and is less dangerous and less costly than blasting, as the expense of maintenance is low.

LIME.

Most marble waste makes excellent lime and a number of quarry companies have constructed batteries of kilns for making lime from the waste. The present and the probable future demand and the price must of course be sufficient to justify the expenditure of the necessary capital for such an enterprise.

FLUX.

If smelting works are situated at a convenient distance so that freight rates are not excessive scrap marble may be sold for fluxing purposes.

IMPROVEMENT OF SOILS.

It is doubtful whether either quarrymen or farmers realize how extensive are the areas of agricultural lands that need liming, or how numerous and diverse are the crops that may be benefited by the application of ground limestone or lime.[a] It is a popular error to assume that soils in limestone or marble areas do not need lime. Many soils that have been formed from limestone, are greatly in need of lime. It is beyond the scope of this bulletin to outline the geographic areas in which lime is needed. Such information may be obtained from the various soil survey reports issued by the Department of Agriculture or by State bureaus carrying on experimental work of a like nature. These reports indicate that large areas of farming lands situated in marble regions are giving mediocre returns, although piles of material that would greatly increase the productiveness of the soil lie unused in the near vicinity.

Government reports state that a judicious liming of the soil would increase by a varying percentage the agricultural productiveness of nearly every State. Consequently, those quarrymen who have vast heaps of waste marble in convenient localities for distribution should attempt a campaign of publicity along such lines as will open this market for their waste products and develop an industry mutually beneficial to both quarrymen and farmers.

Marble waste is being used for soil improvement in a few places, but not nearly as extensively as it might be used. Near quarries at Cockeysville, Md., the upper marble beds are decomposed to a fine powder which is sold as fertilizer without grinding or pulverizing. The slush from marble mills is an excellent material for this purpose, and requires no grinding.

Marble for agricultural purposes should be finely pulverized. The general view is that the finer the material the better. It may be, however, that the expense of producing the finer grades may make it more economical to supply a larger amount of coarser material. It may be stated that at least 80 per cent of the ground rock should pass through a sieve having 100 meshes to the linear inch.

Lime derived from burning limestone or marble in kilns is more effective in the soil than the pulverized rock. Also transportation costs less, as marble loses greatly in weight when burned, although no calcium is lost. There is the same amount of calcium in 2,000 pounds of quicklime as in 3,550 pounds of marble. In other words, 2 tons of pulverized limestone have approximately the same effect on the soil as 1 ton of quicklime. Thus it may be seen that by converting limestone to lime half the freight expense could be eliminated and a more efficient product placed in the market. With the develop-

[a] See Wheeler, H. J., The liming of soils: U. S. Dept. Agr., Farmers' Bull. 77, 1905, 23 pp; the maintenance of fertility; liming the land: Ohio Exp. Sta., Bull. 279, July, 1914, 22 pp.

ment of a wide and active demand for lime fertilizer it may pay to burn the waste marble into lime rather than to pulverize it. Quarrymen who now burn lime from their waste marble should endeavor to develop a market for all waste lime as fertilizer.

TERRAZZO.

For terrazzo flooring the crushed rock fragments should be somewhat cubical in shape and there should be a minimum of material in the form of flat flakes or slivers. Before installing a crusher for producing terrazzo, experiments should be made to determine the type of crusher that will give the maximum proportion of material of the desired form. Fine-grained white marble is in greatest demand for terrazzo, though there is a limited demand for black or other dark colors for making borders or patterns.

CEMENT.

Although marble waste may under favorable conditions be used for the manufacture of cement, its use for this purpose is not usually practicable. The supply is inadequate for the manufacture of cement on a profitable scale. Only where cement plants are situated near marble quarries, and can use the marble waste to supplement their limestone supply, can such a means of utilization be profitably employed.

ROAD MATERIAL.

Crushed marble alone makes a good though not a very durable road. It pulverizes easily and, unless oil or asphaltic material is used, blows away as dust. When mixed with clay it is more durable. It can be used to advantage if more permanent materials are not available.

RUBBLE.

A limited amount of marble is used for foundation stone and retaining walls. Considerable labor is required to reduce the masses to convenient size and shape. The demand is in most places merely local and therefore limited.

MISCELLANEOUS.

Inferior waste material may be used to advantage for local improvement of quarry property. Hollows may be filled and the yard brought to a general level, new roadways may be constructed, or dams built to form water reservoirs.

Certain quarries that use buckwheat coal for fuel, and are situated near large cities, sell their cinders to be used in place of gravel for concrete construction. This is a good illustration of the profitable use of material that many quarrymen would consider to be absolute waste.

STRAIN BREAKS IN QUARRIES.

LOSSES DUE TO STRAIN.

In many quarries in rocks of both sedimentary and igneous origin, at certain stages in the process of rock removal, expansion of the rock will take place, or irregular fractures will suddenly form, occasionally accompanied by subterranean noises. Such minor earth fracturing has been studied by scientists in an effort to obtain new light on earthquake problems. The economic aspects of the phenomena have been almost disregarded, although to quarrymen they are of prime importance. In many quarries a modification of methods may become necessary owing to the interference of strain with the successful operation of certain quarry tools. Of still greater importance is the fact that in a number of marble quarries the formation of the so-called "strain breaks" results in the waste of much valuable material. In recent years it has become necessary to abandon at least three marble quarries in widely separated localities because the marble which otherwise would have been valuable was rendered useless by excessive strain breaks. Such abandonment has occasioned the loss of thousands of dollars, and the excessive waste from the same cause in many other quarries now being worked has resulted in the loss of many thousands more.

STRAIN CAUSES WASTE, NOT IMPAIRED QUALITY.

As reference is subsequently made to particular quarries, one point must be emphasized in order that marble dealers or others may have no misconceptions concerning the effect of "strain" on finished marble. So far as is known, the so-called "condition of strain" in a quarry in no wise detracts from the value of the finished product either in strength or durability. Whatever damage strain breaks cause to the rock is accomplished before the blocks leave the quarry floor and manifests itself in fracturing and rending masses into irregular forms which can not be worked to advantage. The strain causes waste and not impaired quality.

EXTENT AND IMPORTANCE OF STRAIN.

The occurrence of strain breaks is more general than was at first supposed. In view of the heavy losses sustained by several quarry companies, an inquiry into the manifestations and probable origin of strain breaks has been undertaken, and as a result of such investigations new quarry methods are suggested tending toward a reduction in the proportion of waste from this cause. Various instances of strain phenomena recorded in literature are given in brief abstract in chronological order, and to these are added examples from the writer's personal observations.

EXAMPLES OF STRAIN PHENOMENA RECORDED IN LITERATURE.

One of the earliest references to the phenomenon is given in "The Miners Dictionary," by William Hooson, 1747, in which several instances are given of subterranean noises which have greatly alarmed workmen. No attempt at explanation is given. The author states, "Miners say that the knocking is some Being that Inhabits in the Concaves and Hollows of the Earth; and that it is thus kind to some men of suitable tempers, and directs them to the Ore by such its knocking."

In 1854 Johnson[a] described strain in the Portland sandstone quarries near Middletown, Conn. If a uniform channel was cut across the beds in an east-west direction, when the opening reached a point near the bottom of a bed, pressure acting in a north-south direction would suddenly crush into fragments the remaining part. An expansion in the rock mass of three-fourths of an inch is mentioned, but no reference is made to the length of the mass that thus expanded. When a bed was channeled north and south no such compression could be observed. The author states that the rocks are "not perfectly at ease in their ancient bed," but attempts no explanation.

In 1871 Niles[b] described movements and expansions of gneiss as noted in a quarry near Monson, Mass. The strike of the planes of foliation is given as N. 10° E. and the dip as N. 80° W. at an angle of 80°. Cross joints are rare. The rock appears to be under compression in the direction of strike. It was found that fracturing could be prevented by channeling the rock across the strike. A mass of rock 354 feet in length expanded 1½ inches when broken loose. The formation of fractures was accompanied by loud reports, and the projection of dust and rock fragments into the air was noted.

These phenomena are later described by the same author[c] in greater detail. Additional evidence is presented that the rock discussed is clearly under compression and, when freed, expands.

Three years later Niles[d] described a condition of strain in various localities and deduced certain conclusions from these occurrences. The records are briefly summarized in the following paragraphs.

Niles states that in the sandstone quarries of Berea, Ohio, there is evidence of a strong horizontal compressive stress in a north-south direction but none east and west. The process of quarrying requires

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[a] Johnson, John, Notice of some spontaneous movements occasionally observed in the sandstone strata in one of the quarries at Portland, Conn.: Proc. Am. Assn. Adv. Sci., vol. 8, 1854, p. 283.

[b] Niles, W. H., Some interesting phenomena observed in quarrying: Proc. Bos. Soc. Nat. Hist., vol. 14, 1871, p. 80.

[c] Niles, W. H., On some expansions, movements, and fractures of rocks observed at Monson, Mass.: Proc. Am. Assn. Adv. Sci., vol. 22, 1873, pt. 2, p. 156.

[d] Niles, W. H., The geological agency of lateral pressure exhibited by certain movements of rocks: Proc. Bos. Soc. Nat. Hist., vol. 18, 1875–76, p. 272.

channels to be cut at intervals. In quarries with a northerly and southerly working face the channels are cut east and west. When a bed has been channeled nearly through, the stone at the bottom of the channel cut is broken and crushed by the lateral pressure, and the width of the channel is perceptibly decreased by expansion of the adjacent rock. As such crushing renders a portion of the stone worthless for structural purposes, it was found advisable to cut the trenches in short sections in order that the crushing should be confined to limited masses. The expansion and fracturing has been noted at all seasons of the year and hence is seemingly independent of temperature changes. The belief has been general that the pressure was produced by the weight of the adjacent overlying rock. The author points out, however, that the lateral compression observed could not result from vertical pressure on adjacent parts of the beds.

In the Lemont (Ill.) limestone quarries it was noted that in certain places where potholes crossed the boundary between adjacent beds, as a result of the slipping of one bed on the other the upper parts of the holes were offset with respect to the lower parts. An arching of the rock in an anticlinal fold with east-and-west axis was also observed. This again indicated compression in a north-and-south direction. These movements do not perceptibly vary with temperature changes.

In a granite gneiss quarry at Waterford, Conn., where drill holes were made very near each other, the intervening parts of rock were often crushed, and, by expansion of the rock, the drill was so pinched that it could not be operated. The compressive force was in a northeast-and-southwest direction.

Evidences of compression were also noted in quarries at Groton, Conn.

The number and distribution of these occurrences lead the author to conclude that they are due, not to local or external forces nor to peculiarities of composition or metamorphism of the rock mass itself, but to "the continued action of the same geological power which has been the chief agency in the elevation of continents and mountain systems." He likens these fractures and movements to earthquake shocks on a small scale. The tendency toward a north-and-south direction of compressive stress points to a great continental stress in this direction. The gentle undulations that may be noted in the alternate elevation and subsidence of the Atlantic coast from Greenland to Florida are in his opinion the evidence of slight foldings resulting from this compression.

Gilbert [a] discovered several postglacial anticlinal ridges in the horizontal limestones of Jefferson County, N. Y., and in the slates near

[a] Gilbert, G. K.. An account of some new geologic wrinkles: Am. Jour. Sci., 3d ser., vol. 32, 1886, p. 324.

Dunkirk, in western New York. He attributed the expansion that caused such arching to the warming up of the surface rocks as they recovered from the cold of the glacial period.

Strahan [a] describes what he terms "explosive slickensides" in certain lead mines of Derbyshire, England. The walls of the lead veins are rubbed and polished as though moved while in violent contact. When struck or scratched with a pick the wall in many places will break or explode with violence, throwing blocks to a considerable distance. Undercutting of the vein wall has the same effect. His conclusion is as follows: "The explanation which perhaps best satisfies the requirements of the problem appears to be that the spars are in a state of molecular strain, resembling that of the Rupert's drop, or of tempered glass, and that this condition of strain is the result of the earth movements which produced the slickensides."

Hughes [b] refers to similar phenomena in a limestone quarry at Dent Head, and a tunnel at Ribble Head, in Yorkshire, England. He attributes the fracturing to the fact that the limestone rested on shale. When an excavation was made leaving only a thin layer of limestone over the shale, the weight of the overlying limestone on either side was sufficient to cause the shale to flow, after the manner of a viscous fluid, toward the region of reduced pressure. This caused the rock to arch up in the bottom of the quarry. When the brittle rock was thus arched it would break and fly when further excavation was attempted.

Gresley [c] describes explosive fracturing of coal in the mines of South Staffordshire, England. The explosions are termed "bumps," and are said to be of frequent occurrence during the process of excavation, causing many tons of coal to fall down at the working face. Various subterranean rumblings have also been noted. He attributes such phenomena to "the upsetting by the excavation of the equilibrium of the strains or pressure holding everything fast and firm together—the removal of the support, thereby causing the rocks to get relief and to fly off or apparently to explode." He terms such explosions as "miniature earthquakes."

Cramer [d] describes an interesting anticlinal uplift in the bed of the Lower Fox River near Appleton, Wis., which took place immediately beneath a long stone mill, resulting in an arching of the floor and roof with consequent displacement of machinery and cracking of the walls. Two miles distant, at Kaukauna, Wis., the rock between two parallel joints 30 feet apart became arched as a result of compressive stresses. The rock in a zone about 2 feet wide on each side

[a] Strahan, Aubrey, On explosive slickensides. Geol. Mag., vol. 4, Dec. 3, 1887, p. 400.
[b] Hughes, T. McK., Bursting rock surfaces, Geol. Mag., vol. 4, Dec. 3, 1887, p. 511.
[c] Gresley, W. S., Re "Explosive slickensides:" Geol. Mag., vol. 4, Dec. 3, 1887, p. 522.
[d] Cramer, Frank, On recent rock flexure: Amer. Jour. Sci., ser. 3, vol. 39, 1890, p. 220.

of the joints was splintered by the pressure. No explanation is presented, though the author refers to Gilbert's suggestion that such phenomena may be due to expansion caused by a rise in temperature after the disappearance of ice from glacial regions.

Reade[a] discusses various references to strain. He cites many instances of recent changes of level in the surface of the earth and concludes that from "the prevalence of these vertical movements in recent geologic times it is obvious that the subsidence of a low arch of elevation must tend to put the surface rocks into lateral compression." This state of compression may be influenced by the presence of joints and faults. The fact that conditions of lateral compression are recorded as unusual raises a strong presumption that tangential thrust arising from a shrinkage of the earth's nucleus is not the agent. He concludes therefore that the phenomena are due to the changes of level mentioned above.

A more complete examination of the anticlinal arch at Appleton, Wis., was obtained when the water was removed from part of it and a further description was given by Cramer[b] in 1891. He records several other instances of strain breaks in quarries of that vicinity. The direction of fracturing is highly variable, and seems to be determined rather by the means of relief than by the preponderance of pressure. He concludes that the rock is under compression "in all directions." A set of anticlinal ridges parallel with the strike of the Corniferous limestone beds at Lime Rock in Genesee County, N. Y., are, however, attributed to rock collapse produced by solution and removal of salt and gypsum from the underlying beds.

Gilbert[c] describes anticlinal ridges 6 to 8 feet high in the Devonian shale of Ripley township, in western New York. He notes similar anticlinals in the Devonian shale of northwestern Ohio and in the Trenton limestone in northern New York. He reiterates his former contention that all are to be attributed to expansion due to postglacial rise in temperature.

Matthew[d] describes numerous thrust faults in the slates near St. John, New Brunswick, the most probable cause of which he gives as lateral thrust from the southeast.

Chalmers[e] describes numerous thrust faults in the slates of southeastern Quebec. They all occur near mountains or resisting masses

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[a] Reade, T. M., The cause of active compressive stress in rocks and recent rock flexures: Amer. Jour. Sci., ser. 3, vol. 41, 1891, p. 409.

[b] Cramer, Frank, On the rock fracture at the combined locks mill, Appleton, Wis.: Am. Jour. Sci., ser. 3, vol. 41, 1891, pp. 432-434.

[c] Gilbert, G. K., Postglacial anticlinal ridges near Ripley and Caledonia, N. Y.: Am. Geol., vol. 8, 1891, pp. 230-231.

[d] Matthew, G. F., Movements of the earth's crust at St. John, New Brunswick, in postglacial times: New Brunswick Nat. Hist. Soc. Bull. 12, 1894, pp. 34-42.

[e] Chalmers, R., Report on the surface geology and auriferous deposits of southeastern Quebec: Geol. Survey of Can. Ann. Rep., vol. 10, 1898, pt. J, pp. 9-12.

of rock and are attributed to a pushing of the beds against such masses, to a sinking of the resisting masses by cooling and contraction, or to both.

Campbell [a] has described a fold in sandstone which he attributes to processes of weathering. The increase in volume by weathering causes a forcing apart of joints and cleavage fissures. The process is assisted by thermal expansion and contraction by freezing of water in the joints and by the force of growing roots. The cumulative effect of such forces in many joints produces rupture and arching at a point of weakness.

Beard [b] intimates that gas escapes into coal mines as a result of earth movement, and in this connection shows that many mine explosions are contemporaneous with earthquake shocks.

Woodworth [c] states that thrust faults of recent date occurred near Troy, N. Y. He notes several brick and stone structures resting on the rock, which are fractured and faulted, showing that faulting has occurred since the structures were built. Similar faults were noted in Rensselaer, Defreestville, and Pumpkin Hollow. The whole series is attributed to a mountain-building thrust similar to and in the same direction as that which formed the mountains of the region. Reference is made also to postglacial faults in Quebec, Massachusetts, and New Hampshire.

Dale [d] describes a state of compression in granite quarries of Waldo, Hancock, Kennebec, and Lincoln Counties, Me. He also cites the observations of Gilbert at Lithonia, Ga., and reproduces a photograph of an anticlinal arch formed by compressive stresses in the granite of this region. The same author [e] describes the arching of Vermont granite under compression. Under a later date he describes [f] in detail the compressive stresses in various Vermont granites, and gives the direction of compression of each. At Bethel the direction was east-west, at Barre, chiefly north-south, at Woodbury, northeast-southwest, at Groton, in all directions, at Ryegate, east-west, at Dummerston, N. 10° E. to S. 10° W. The effect of the strain is to close channels and crush cores between drill holes.

Van Horn [g] refers to a series of anticlines in the Chagrin shales of Cleveland, Ohio, which he attributes to the increase in volume that results when iron sulphide contained in the shales alters to iron sulphate and alum-like compounds.

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[a] Campbell, D. F., Rock folds due to weathering: Jour. Geol., vol. 14, 1906, pp. 718–721.

[b] Beard, J. T., Colliery explosions and their causes: Eng. and Min. Jour., vol. 83, 1907, pp. 1051–55.

[c] Woodworth, J. B., Postglacial faults of eastern New York: New York State Museum Bull. 107, 1907, pp. 5–28.

[d] Dale, T. N., The granites of Maine: U. S. Geol. Survey Bull. 313, 1907, pp. 34, 42, 121, 142.

[e] Dale, T. N., Chief commercial granites of Massachusetts: U. S. Geol. Survey Bull. 354, 1908, p. 25, B.

[f] Dale, T. N., The granites of Vermont: U. S. Geol. Survey Bull. 404, 1910, pp. 17–18.

[g] Van Horn, F. R., Local anticlines in the Chagrin shales at Cleveland, Ohio: Geol. Soc. Am. Bull., vol. 21, 1910, pp. 771–773.

Lawson [a] describes a series of postglacial thrust faults in Archean slates near Banning, Ont. He attributes these and similar thrust faults to volume change in the surface rocks brought about by chemical or temperature changes.

EXAMPLES OF STRAIN PHENOMENA OBSERVED IN FIELD WORK.

Other examples of strain phenomena noted by the writer during the field season of 1914 are described below.

In a quarry near Knoxville, Tenn., the strike of the marble beds is N. 55° E., and the dip 35° to 40° SE. Slip joints appear parallel or nearly parallel with the bedding. In certain parts of the quarry zones of parallel joints run N. 40° to 50° W. The rock is subjected to a severe compressive stress approximately parallel with the strike. Great difficulty is experienced in separating the blocks without irregular fractures. Much valuable stone is lost by fracturing into angular pieces during the process of quarrying. A distinct though small earthquake shock is reported by a quarryman to have occurred a few years ago within half a mile of this quarry. This was probably only another manifestation of the condition of strain within the rock. In this instance it exceeded the elastic limit, and sudden fracture and displacement occurred.

A blue-marble quarry situated about a mile and a half north of Florence, Vt., has had some trouble with strain breaks though no details were procurable. Other quarries near the same locality have suffered severely from the same cause. One quarry was abandoned because the strain breaks destroyed so much rock that excavation could not be conducted with profit. The rock is above the average in soundness, masses 35 feet in length having been quarried for columns.

A quarry near Clarendon, Vt., had to be abandoned on account of excessive strain breaks. When the author visited the quarry the pit was full of water. The structure as described by Dale [b] is as follows: The beds strike N. 10° W. and dip 42° W. At least one bed is crossed by a slip cleavage. Joints strike N. 35° W., and dip 45° to 55° W. The rock has a distinct rift parallel with the bedding. When an opening of considerable depth was made near the eastern end of the excavation a violent fracture occurred which shook houses in the vicinity and frightened the workmen so that they could not be induced to reenter the pit for several days. The fractures took place toward the west and thus ran downward on the rift, destroying the rock to such an extent that the pit was abandoned.

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[a] Lawson, A. C., On some postglacial faults near Banning, Ont.; Seism. Soc. Amer. Bull., vol. 1, December, 1911, p. 159.

[b] Dale, T. N., The commercial marbles of western Vermont: U. S. Geol. Survey Bull. 521, 1912, p. 112.

A quarry not far distant was affected by similar, though less extensive, strain breaks. In this quarry the beds strike N. 55° to 60° E., and dip 25° N. 33° W. They are crossed by cleavage dipping 25° E. The direction of strain was not ascertained.

The granite at Vinal Haven, Me., lies in sheets 2 to 5 feet thick. Vertical joints are rare. In quarrying out a floor it has been found that if all material is removed from the upper surface of a sheet 3 feet thick and 50 feet long, the sheet will arch at least three-fourths of an inch in the center. When one end is set free by channeling it will settle back into its original position.

In a quarry near Woodbury Center, Vt., a condition of strain resulted in a minor accident. The projection of a row of horizontal drill holes formed a plane of weakness, and owing to internal strain a mass of rock broke lose and struck a driller's hand with sufficient force to cause a flesh wound.

At Stonington, Me., in certain granite quarries the channel-bar method can not be employed to obtain a heading, as when holes are drilled close together the strain is sufficient to crush the parts of rock between the holes and to jam the drills. To avoid this difficulty a 2-foot channel is made by drilling and blasting. It is stated that when a mass of rock 50 feet long is set free at one end it will stretch three-fourths of an inch.

At Lithonia, Ga., the granite forms anticlinal arches by compression. One of these arches observed by the author was about 30 feet across and was raised so as to leave an open space beneath about 14 inches high.

GENERAL ANALYSIS OF STRAIN BREAKS RECORDED.

An analysis of all the examples noted and the explanations given by various authors points to the conclusion that by far the greater majority of all serious strain breaks in quarries are in reality minor earthquakes and must therefore be attributed to the same causes as earthquakes. The case recorded above of a strain break in a Vermont marble quarry of such intensity that it shook structures in the immediate vicinity is a striking example of the identity of strain breaks with earthquakes.

It seems probable that strain breaks in general are not to be attributed to local causes but to great and far-reaching forces. Most of the explanations given above are in accordance with this hypothesis. Explanations opposed to the hypothesis are as follows:

Both Gilbert and Lawson give as a probable explanation a rise in temperature following the removal of glacial ice. Although this is a possible cause, and may be contributory, its acceptance as the chief cause is discouraged by the fact that in regions in Georgia and

Tennessee that are beyond the limits of glaciation are strain breaks seemingly similar to those in New England.

Hughes's theory of rock flowage in a plastic layer beneath a limestone bed is of local application only, as in most stone quarries no such plastic layer exists.

Cramer's theory of displacement by collapse due to subterranean solution is also of local application only.

The hypotheses of Campbell, of Lawson, and of Van Horn that buckling is due to expansion as a result of the chemical or physical changes which accompany weathering may apply to a few localized superficial strains, but not to a condition of strain at depth.

In addition to the hypotheses mentioned above it seems probable that temperature changes may cause local expansion of rock, as for example the arching of granite, which has been noted at Lithonia, Ga. Merrill [a] refers to the results of tests made at the Watertown Arsenal which indicate that rocks do not completely recover after expansion, but assume a permanent "set." Repeated expansions and contractions by diurnal or seasonal changes in temperature may therefore cause a gradual expansion of surface layers with consequent arching. Changes of temperature are slight at a short distance beneath the surface, and consequently it is unlikely that a state of compression deep within quarries can be attributed to this cause.

All the other explanations offered in the preceding pages have a more or less direct bearing on the theory that the majority of strain breaks are due to the great forces of nature that build mountains and continents, and produce earthquakes.

Dale [b] states that the cause of the strain observed in certain eastern Tennessee marble quarries is the same as that which produced the Appalachian folds, and which operated both parallel and at right angles to the axes of the folds, the former only being operative at present.

RELATION OF EARTHQUAKES TO STRAIN BREAKS.

Milne [c] divides earthquakes into two groups, the macroseismic or great disturbances, and the microseismic or minor shocks. The latter are described as "settlements and adjustments along lines of their (the greater earthquake's) primary fractures," and are confined to areas of a few miles' radius. He estimates the number of the latter as at least 30,000 a year for the entire earth.

Hobbs [d] has pointed out that constant adjustments and changes of level are taking place in the earth's crust, as evidenced by post-

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[a] Merrill, G. P., Stones for building and decoration, 1903, p. 478.

[b] Dale, T. N., The marbles of eastern Tennessee, unpublished manuscript.

[c] Milne, J., Seismological observations and earth physics: Geog. Jour., vol. 21, 1903, p. 1.

[d] Hobbs, W. H., Earthquakes, 1907, pp. 211-223.

glacial faults and known variations in altitude of certain geographic areas, with respect to sea level. Such gradual changes are termed "bradysisms." He refers to the common occurrence of subterranean rumblings, "brontidi," which are not accompanied by sensible earthquake shocks. These are attributed to slow adjustments of the earth's crust, such adjustments taking place as frequently repeated slight displacements rather than as gradual movements.

Chamberlain [a] claims that the earth acts like an elastic, rigid, crystalline body in which strains accumulate until they reach an intensive stage, and then yield either to slow movements of great magnitude, as illustrated by various changes in level, or by sudden, swift, noncontinuous movements of less magnitude, as earthquakes.

Gilbert [b] states that earthquakes are of two types, as follows: (1) The tectonic, those arising from subterranean mountain building forces, and (2) the volcanic, those resulting from the movements of lavas. The former are the more important, especially in the United States. The tectonic process is briefly described by him as follows:

> In the formation of mountains and other great features of the earth the rock masses are forced into new shapes. They are pulled, pushed, twisted, and bent, so that strata, for example, which were originally flat, become inclined and curved. If the changes are sufficiently slow, the component particles of the rock readjust themselves gradually; but if the changes are comparatively rapid, the rocks are broken. Before fracture occurs there is elastic yielding or "strain;" that is, the rock is compressed or stretched or bent somewhat like a spring; and when its strength is at last overcome the dissevered parts recoil. This recoil is instantaneous, violent, and powerful, and is in the nature of a jar.

As a result of observations of earthquakes in Panama, MacDonald [c] concludes that they are manifestations of relief from strain. The noises resembling ice fractures on a cold morning appeared to be "due to the formation of small shears or strain-relieving cracks in the rocks."

ORIGIN OF THE STRESSES.

Deformation of the earth's crust is constantly taking place. Acting under the force of gravity the earth's shell suffers collapse and buckling. According to the isostatic hypothesis it is assumed that the continents with their plateaus and mountains are composed of and deeply underlaid by lighter matter than the sea bottoms and therefore stand higher because they float higher. A variation in load due to erosion and deposition causes an undercurrent or flow of rock material from the area loaded toward the area denuded. Opposed to the isostatic theory is that of high rigidity within the earth's crust,

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[a] Chamberlain, T. C., Diastrophism and the formative processes: Jour. Geol., vol. 21, November-December, 1913, p. 681.

[b] Gilbert, G. K., The cause and nature of earthquakes: Min. Sci. Press, vol. 92, April 28, 1906, p. 272.

[c] MacDonald, D. F., Some earthquake phenomena noted in Panama: Science, new series, vol. 41, May, 1915, p. 783.

a rigidity sufficient to sustain mountains and continents in higher position than sea bottoms merely by lateral thrust.

According to the former hypothesis the flow of rock from one point to another is resisted by rock rigidity, and a condition of internal strain results. Dutton [a] states that one of the probable causes of many earthquakes is the tendency of the earth to recover its isostatic equilibrium when, by denudation, transportation, and deposition of vast masses of material, this condition of equilibrium is destroyed. The theory of high rigidity, on the other hand, assumes that thrust rather than flow causes internal strain. Whatever may be the correct theory the fact remains that changes of level are of constant occurrence, and strains varying in intensity and direction are the result of such adjustments.

ACTION OF ROCKS UNDER STRESSES.

In order to describe the behavior of rocks under stress some principles of earth physics must be emphasized. Hoskins [b] states that "a body is strained when the relative positions of its particles undergo any change." If the limit of elasticity is exceeded, the change will become permanent—that is, the body will not assume its original form when the force is removed. This permanent change of form may find expression in fracture or in flowage, depending upon plasticity. If fracture results, earthquakes of greater or lesser intensity are produced.

When plasticity overcomes elasticity flowage will take place rather than fracture. This condition is illustrated by the somewhat intricate folding in many mountainous regions. Chamberlain [c] has shown that deformation having the aspect of plastic deformation may take place by a process of granulation and progressive recrystallization. Such a process undoubtedly does take place in marble, as evidenced by the extreme recrystallization.

The matter of supreme importance in this discussion is, however, the state and action of the rock under stress within the elastic limit. If the limit of elasticity has not been exceeded the body will recover its natural configuration when the stress is removed. That rocks are both compressible and highly elastic has been clearly shown by the numerous examples of automatic recovery from strain referred to in previous pages, and has been proven in the laboratory by the experiments of Adams and Coker.[d] A condition of strain within the elastic limit is that condition which quarrymen call "strain." The

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[a] Dutton, C. E., Earthquakes, 1904, p. 37.

[b] Hoskins, L. M., Flow and fracture of rocks as related to structure: Sixteenth Ann. Rept. U. S. Geol. Survey, pt. 1, 1896, p. 860.

[c] Chamberlain, T. C., Diastrophism and the formative processes: Jour. Geol., vol. 21, 26, p. 678.

[d] Adams, F. D., and Coker, E. G., An investigation into the elastic constants of rocks, more especially with reference to cubic compressibility: Carnegie Inst., Wash., Pub. 46, 1906, p. 16.

latter term includes also all permanent deformations. For the sake of simplicity the term "strain" is subsequently used in the sense of strain within the elastic limit. This is the condition of rock masses that has led to such disastrous losses in many marble quarries.

The exact relation of elasticity to plasticity of rocks under varying conditions has a direct bearing on the permanence of strain. Rudzki[a] states that the elastic force with which a body resists a deforming force diminishes with time. Hobbs[b] in applying this principle to rock folds concludes that the period of time covered by the evolution of a fold may be sufficiently long to permit the resistance within the mass to fall. If this conclusion is correct, we may expect that after a great lapse of time plasticity will overcome elasticity and equilibrium will be attained by folding without fracture. Folding may thus take place within what is usually termed the zone of fracture if the process is sufficiently gradual. If, however, nature's process hastens the operation beyond a certain "speed limit," fracture must result.

Thus it would appear that after a tremendous lapse of time permanent deformation may take the place of strain, and a state of perfect equilibrium be attained. No strain breaks would occur in quarries situated in such masses. It might be concluded that marble deposits that have not visibly suffered mountain-building movements since Cambrian or other early geologic age would have attained such a state of rest, and would therefore be free from strain.

It is possible that strain is more common in regions where intense folding is of comparatively recent age. The examples of destructive strain in marble so far observed are all to be found in the Appalachian belt, which was folded toward the end of the Paleozoic era. No hard and fast rule can be given, however, because, as already pointed out, ever-varying earth movements may cause internal stresses anywhere and at any time.

Rudzki's principle further states that temperature has a decided influence on plasticity, an increase in temperature resulting in a corresponding increase in plasticity. The effect of overburden is to increase the temperature and therefore to increase the plasticity. Denudation lowers the temperature, decreases the plasticity, and therefore increases the tendency to fracturing rather than to deformation by flowage. In surface rocks, consequently, the relaxation time is great, and a condition of strain may exist ages after the initial force has been exerted if the force tending toward deformation has remained constant.

The foregoing discussion indicates that a strain may be inherited from a former geologic era, possibly accounting for a condition of

[a] Rudzki, M. P., Physik der Erde, Leipzig, 1911, pp. 232–233.
[b] Hobbs, W. H., Mechanics of formation of arcuate mountains: Jour. Geol. vol. 22, 1914, pp. 193–194.

strain in some isolated rock hills to which it seems impossible to some scientists that present forces can transmit their effects. It is conceivable also that rocks now at the surface were formerly deeply buried and subjected to such intense pressure from superincumbent material that they were forced to assume a smaller volume. Processes of denudation gradually brought these masses nearer to the surface, where the residual strain might find expression in shock fracture, or where a means of relief in the form of strain breaks was obtained through quarry operations.

THE LIMITATIONS OF STRAIN.

As these forces are so far-reaching and constant, it might be expected that strains would be almost universal. One explanation that they are not is the limited cubic compressibility of rocks. When strain in a granite is relieved, an expansion of three-fourths of an inch over a length of 50 feet is perhaps a fair illustration of the compression possible without rupture. It is evident, therefore, that little lateral movement is required to bring relief. Most rocks are intersected by joint planes, which permit the necessary movement to give relief from pressure. It is significant that quarries having considerable unsoundness have little or no strain. There is, therefore, some consolation in the fact that these two great evils are not likely to occur in the same quarry.

Strain in one direction only has been noted in a number of localities. In some places the great earth stresses may be in one direction only. In others it seems probable that strain is produced because rocks can find no relief in one direction, but are free in all other directions. It is significant that as regards most rocks having a steep dip, the strain is parallel with the strike and no strain is apparent across the strike.[a] A slipping of the beds one upon another may give the necessary relief in the transverse direction, whereas parallel with the strike the absence of joints or planes of weakness may afford no such slipping. In a Tennessee quarry the strain is approximately parallel with the strike and there is evidence of slipping parallel with the beds, which gives the necessary relief in the transverse direction.

EFFECT OF QUARRYING ON STRAIN.

In quarries in which strain exists the internal strain-stress relations may be complex. Various forces are acting and reacting, and their total result is a condition approximating equilibrium. The fact that a condition of strain exists within the limit of elasticity indicates that true equilibrium has not been attained. According to Rudzki's principle, the elastic resistance of the rock is gradually

[a] Dale, T. N., The marbles of eastern Tennessee, unpublished manuscript.

diminishing and slow permanent deformation is constantly going on. As a result of excavation the approximate equilibrium is thrown out of balance locally. If we could imagine all stresses except the downward force of gravity ceasing simultaneously, we could conceive of the strained rock recovering its natural configuration without rupture. Even then the realignment of particles in a thick bed could scarcely take place rapidly without fracture. In the stone quarry the conditions are more extreme. The strain-stress relations are disturbed in a limited region only.

Before excavation is begun a stress that tends to deform the rock is balanced by the resistance of the rock plus the weight or lateral pressure of other rock masses. When these masses are removed the resistance of the rock alone is not sufficient to withstand the deforming stress, and fracture results. This is clearly illustrated by the examples previously given of the crushing of rock at the bottom of channels in the Portland and Berea sandstone quarries (pp. 124–125). Such fractures are the result of compression.

Strain may manifest itself in other ways. At some stage in the process of cutting a block of marble free the support that holds it in compression may no longer be sufficient and the block expands. As it is still rigidly connected with the main rock mass, which remains under compression, no considerable expansion can take place without fracturing between the two masses. Such fracturing may be in the nature of shearing, with an even break between the two masses and lateral displacement.

Thus it is established by theory, which unfortunately has been confirmed by practice in many places, that in quarries where strain exists the successive separation of small masses from the main quarry rock generally results in the formation of fractures, producing excessive waste. Some new method of quarrying must be found if this waste is to be reduced.

BASIC PRINCIPLES LOOKING TO REDUCTION OF DAMAGE FROM STRAINS.

The following basic principles looking to a solution of the problem have been established: (1) The causes of strain are not to be found in mere local conditions such as the weight of a near-by hill or the process of surface weathering. Strains are manifestations of the great geologic forces that build mountains and cause earthquakes. (2) In every observed instance of strain the rock has been under compression. (3) The compression is usually in one definite direction. (4) The only way to remove the strain is to bring about expansion of the rock to its normal volume. (5) If the strain on a small mass of rock is relieved, it expands independently of the main mass, with consequent fracturing; therefore relief from strain should be sought for a large mass at one time.

THE REMEDY.

The principles outlined indicate that some means should be found to cause uniform expansion of a large mass of rock at one time in such a manner as to reduce to a minimum the occurrence of irregular fractures between the mass set free and the main quarry rock. When the strain has been relieved the mass can be quarried without fear of fractures. The first step to be made in quarrying rock under strain is to determine the direction of compression. In most instances quarrying will already have been done, as otherwise the condition of strain will not have been revealed. It is usually easy to determine the direction of compression by noting the direction of movement of the loosened blocks with reference to the main mass. The lateral displacement of drill holes or other marks will indicate the direction.

In order to bring about the necessary relief an opening should be made at right angles to the direction of compression. Expansion of the rock will manifest itself by a gradual narrowing of this space, and the rock will thereby find relief from strain. The opening should be of sufficient width to allow complete relief by expansion and of sufficient length and depth to free a large mass of rock at one time.

A channeling machine would probably give poor results, as an open cut gives too ready a means of relief. The pressure would tend to close the cut immediately and to jam the bits, and only a small mass of rock would be freed. Also the opening should in most instances be deeper than a channeling machine can cut. Drilling a row of close holes would probably afford the best means of relief. The compressive force would gradually crush the rock remaining between the drill holes, close up the holes, and thus relieve the strain. The presence of these supporting cores would prevent a too rapid closing, and would thus allow uninterrupted drilling.

If the direction of compression is parallel or approximately parallel with the quarry walls the quarryman is fortunate. Rows of deep vertical holes should be drilled along both quarry walls parallel with this direction. The holes should be deep enough to pass entirely through the bed if the latter is horizontal or nearly so, and if open or weak bedding planes occur. In massive or steeply inclined beds the holes should be at least 12 feet, and preferably 15 or 20 feet deep. Convenient sizes are 1¾ or 2 inches in diameter. The width of rock intervening between the drill holes should average not more than 1 inch.

When the rows have been completed, a row of holes of similar depth and spacing should be projected in a line at right angles. This third row may be placed at either end of the quarry or in some intermediate position. The latter is probably more effective especially if the compression is not exactly at right angles to the line of drill holes, as motion in opposite directions in both masses will afford a more ready means of the cores slipping on each other.

The purpose of the two lines of holes parallel with the direction of compression is to give an easy shear line when expansion takes place. When they are drilled before the cross line of holes, no motion occurs to jam the drills. If, however, they are drilled after the cross line the tendency of the rock to offset by shearing would continually interfere with drilling operations.

The transverse line of holes should be projected rapidly. Several drills should be employed in order to complete the line as nearly as possible before any appreciable closing of the holes takes place.

Care should be taken that no excessively wide cores are left in the corners. The holes should be well cleaned out and covered in some way to keep out débris. Tightly driven hardwood plugs may tend to prevent the desired closing of the holes. Plugs of cotton waste or similar material may be used.

Along the transverse line a crushing of the cores and a closing of the drill holes will indicate expansion of the rock mass. There will probably be some lateral motion, permitting the cores to slip by each other and project into the open holes.

Along the shear lines expansion of the rock will not appreciably close the holes, but will by lateral motion offset one-half of each hole with respect to the other half. The extent of lateral displacement will diminish with distance from the transverse row of holes.

What may occur at the bottom of a mass of rock that is thus permitted to expand is as yet by no means certain. It is possible that fracturing may take place, and fractures so made will tend to occupy diagonal positions, causing much waste. It is for the purpose of reducing such fractures to a minimum that deep holes are recommended.

If irregular bottom fractures cause excessive waste, or if the expense of making deep holes is considered to be too great, an alternate method is proposed, as follows:

A narrow part of a floor along one side of the quarry parallel with the direction of compression may be first removed. The remainder of the floor may then be quarried as shown in figure 30. A row of horizontal holes along the floor at *a* should be closely spaced. A similar line of vertical holes is then projected at *b*. If the direction of compression is parallel or nearly parallel with the face, it is probable that a channel cut may be substituted for this second line of holes. If now a cross line of holes, *c*, is made at either end of the mass or in some intermediate position, expansion will crush the cores and close the holes. Along the planes *a* and *b* the cores will break and shear. Theoretically, at least, no irregular fractures should occur, as an even plane has been provided on every side. It is impossible to foresee what disturbing elements may arise, but the method holds sufficient promise to justify a trial.

If the direction of compression is diagonal to the quarry walls, it may be necessary to change the directions of channeling and drilling in order that they may more nearly conform with the strain line. This change will lead to a reduction in floor space occasioned by leaving masses in the corners. When the angle between the direction of compression and the direction of a quarry wall is not more than 15° or 20°, channeling and drilling in the usual directions will probably be satisfactory.

FIGURE 30.—Method of drilling holes to avoid strain breaks.

A SUCCESSFUL EXPERIMENT.

A Tennessee marble company, acting on the advice of Mr. T. Nelson Dale, of the United States Geological Survey, who visited the quarry during the summer of 1914, made an effort to relieve strain by drilling holes in accordance with the first plan outlined above. Although defects were apparent in the preliminary trial, the great increase in the proportion of sound stock produced encouraged a continuation of the method. During April, 1915, the writer observed the progress that had been made up to that time and offered some suggestions toward further improving the method. A complete description of the plan of drilling, effect on the rock, and of suggested changes in method follows.

The mass of rock under observation was 69 feet long, 20 feet wide, and 12 feet thick. It occupied a position along one side of the quarry, the remainder of the floor to the northwest having already been removed. Its position is shown at *A* in figure 31.

A line of holes perpendicular to the supposed direction of compression was first drilled. The holes were 1¾ inches in diameter, 12 feet deep, and the average core thickness between drill holes was fifteen-sixteenths of an inch. The direction of the line was N. 40° W. The line was 20 feet long, extending from the corner of the quarry along the

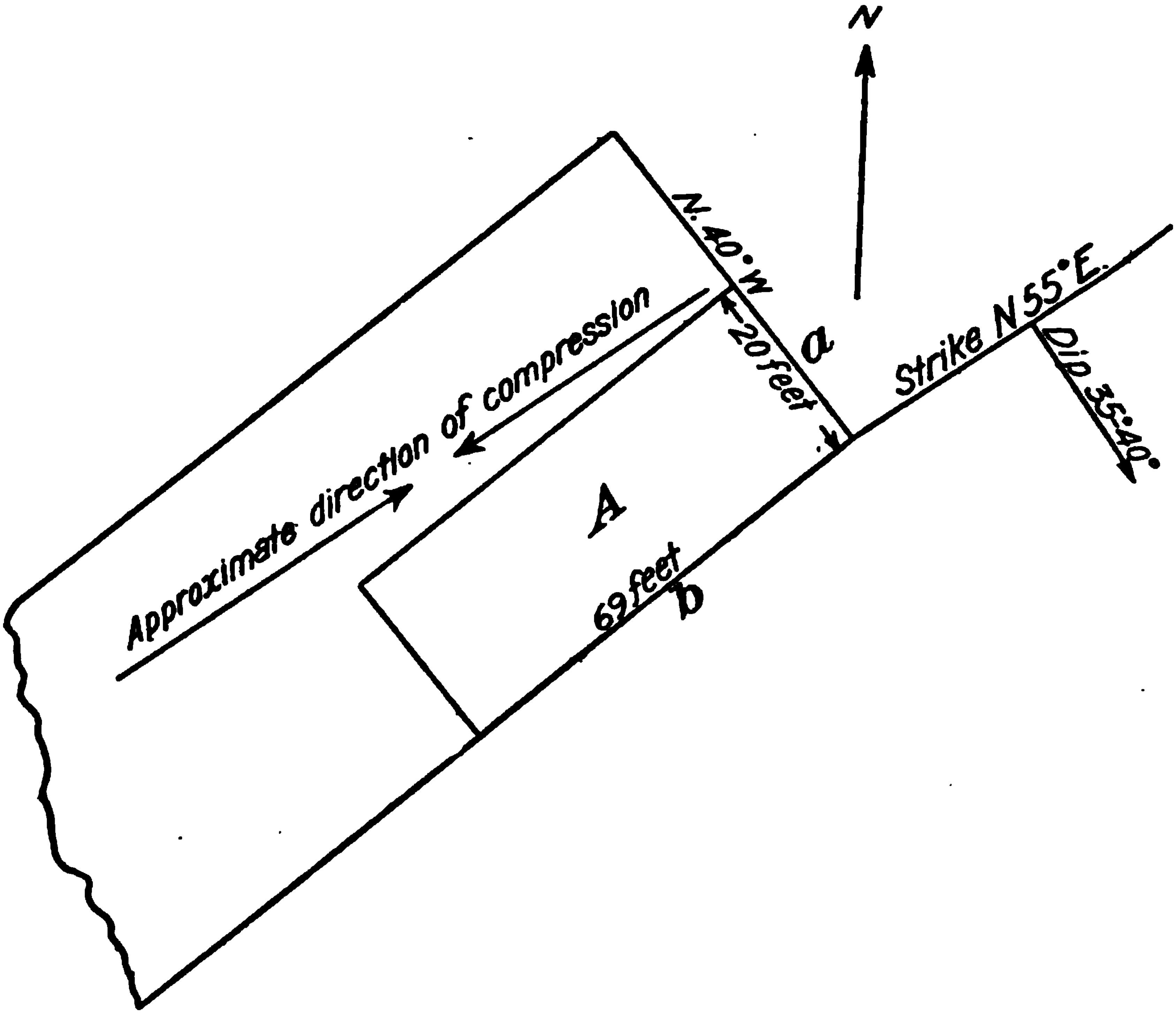

FIGURE 31.—Plan of part of a marble quarry, showing position of a rock mass studied. A, rock mass where observations were made; *a*, position of line of holes driven perpendicular to supposed direction of compression; *b*, position of second line of holes driven to give rock mass freedom to shear on expanding.

northeastern wall and terminating at the excavated part of the floor. Its position is indicated by *a* in figure 31.

A second line of holes of similar depth and spacing was projected along the southeastern wall of the quarry, as shown at *b* in figure 31. The purpose of this line of drill holes was to give the mass of rock freedom to shear when it expanded. On account of the difficulty of placing drills, holes were omitted for a distance of 10 inches from the corner on the northeast wall, and 6 inches from the corner on the southeast wall. As indicated in figure 32, a mass of rock 12 inches wide was left in the corner.

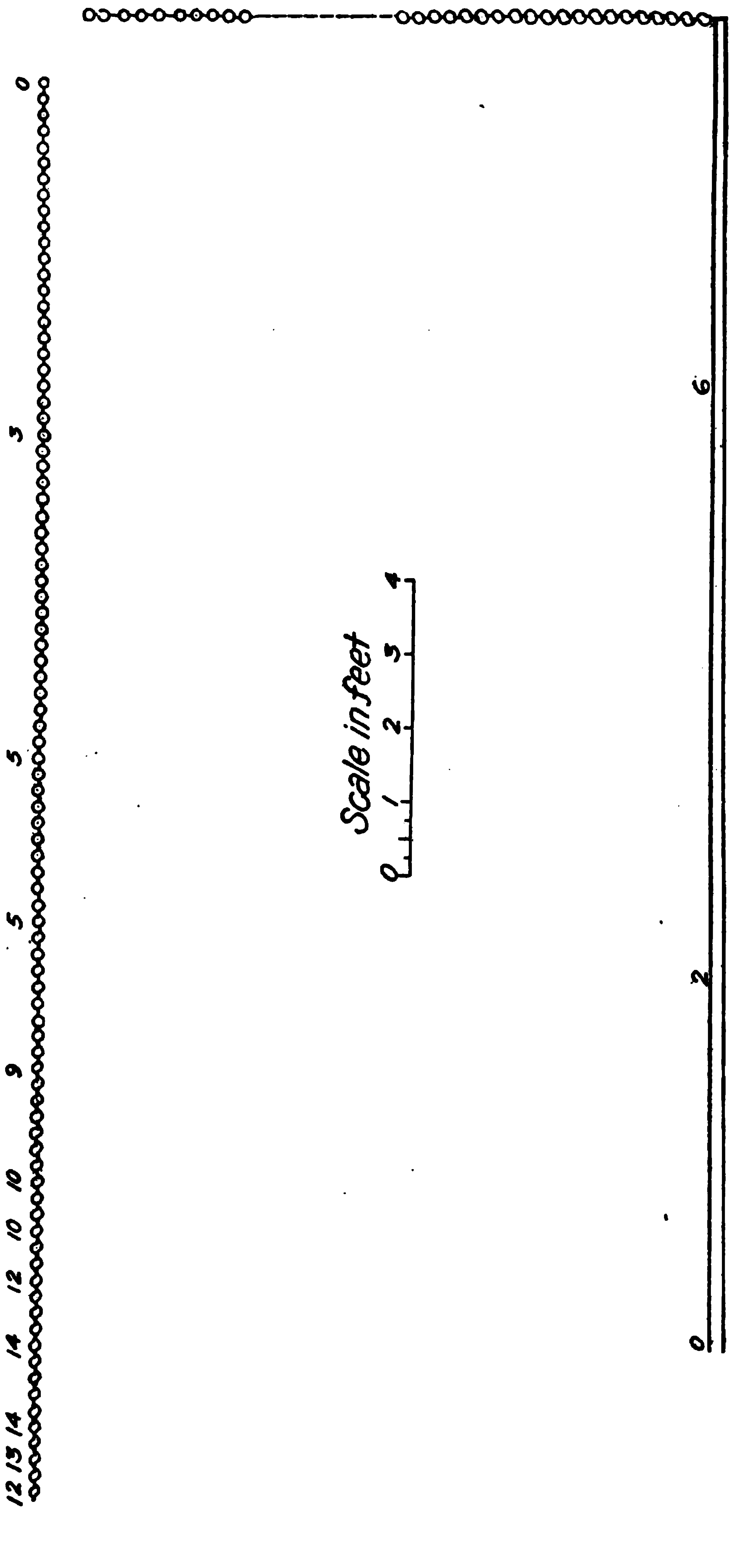

FIGURE 32.—Appearance of drill holes and channel cut after expansion of rock. Size of drill holes and their shape after expansion are drawn accurately to scale. The numbers indicate the degree of closing, in thirty-seconds of an inch, at various points.

Expansion of the marble soon manifested itself by a crushing of the cores and a gradual closing of the drill holes. The closing did not take place uniformly. Near the northwestern end of the line it was greatest, attaining a maximum of seven-sixteenths of an inch, which gradually decreased to zero at the other end of the line. There were 88 drill holes in the 20-foot line. The extent of closing as measured at various points was as follows:

Hole No.	1	3	5	10	14	15	17	20	27	36	64	88
Extent of closing in thirty-seconds of an inch	12	13	14	14	12	11	10	10	9	5	3	0

The appearance of the partly closed holes is shown in Plate XII, *B*. The manner in which expansion took place is shown in figure 32. The cores were broken the entire length of the line, though the movement in the last holes was too slight to be recognizable with the naked eye. Likewise on the perpendicular line every core was broken by shearing. However, the mass in the corner was unbroken, as far as could be seen, and it would therefore appear that this mass of rock,

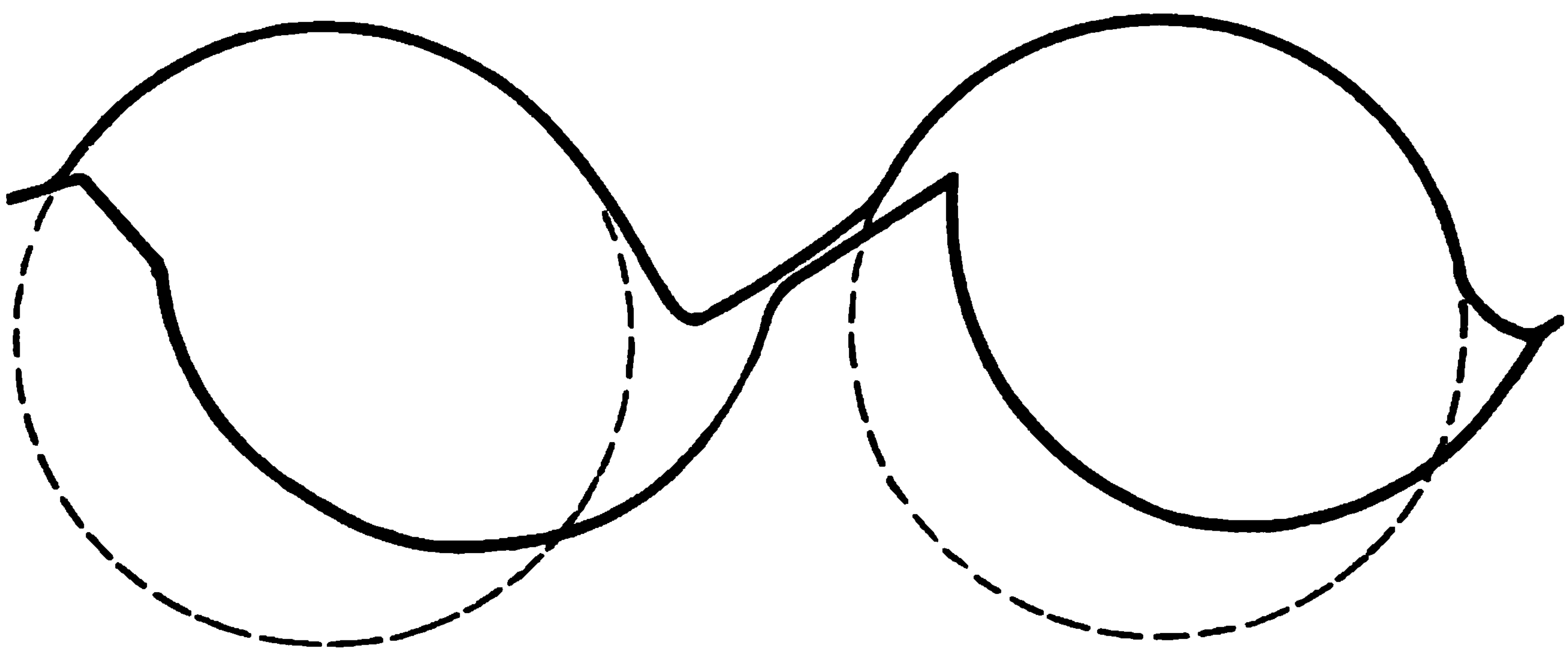

FIGURE 33.—Second and third drill holes from the left end of the line shown in figure 32, illustrating the manner in which the holes were closed by compression. Actual size and shape; dotted lines show original position of drill-hole walls; solid lines show position after rock movement.

which was 12 inches in width, effectively prevented the major part of the mass to expand so as to gain relief.

Figure 33 illustrates the second and third drill holes in the cross line drawn natural size. In the lower part of the sketch the light line represents the original position of the drill hole walls, and the heavy line the position of the same walls subsequent to the expanding movement. It is evident that motion took place in a diagonal direction, which is undoubtedly due, in part at least, to a lateral slipping of the cores in order that they might gain ready relief by projecting into the open holes. However, the fact that slipping took place toward the corner where a mass of solid rock prevented any movement would seem to indicate that the direction of compression was not

exactly perpendicular to the line of drill holes, but inclined at a small angle toward this corner.

At a later time a channel cut was made parallel to the first line of holes and 69 feet distant from it as indicated in figure 32. After the cut had been completed, the fact that complete expansion had not occurred along the first line of drill holes became apparent. That part of the 69-foot mass that had previously been unable to expand eastward on account of the uncut corner began to expand westward and partly closed the channel cut. The shearing accompanying this expansion broke the cores between the drill holes along the entire shear line, and the displacement gradually increased with the distance from the corner.

Measurements were made of the closing of the channel cut. Owing to a shattered condition of the rock, measurements could not be made nearer than 5 feet to the quarry wall. At this point the cut was closed six thirty-seconds of an inch, 13 feet from the wall it was closed two thirty-seconds of an inch, and 18 feet from the wall it was not closed at all.

Thus it is evident that where relief was gained along the original line of drill holes no further relief was necessary, but where in the original line no motion was possible a backward motion took place as soon as a means of relief was provided.

A considerable number of strain breaks are present in this mass of rock. It is reasonable to assume that they were formed in consequence of the nonuniform and imperfect relief obtained, as pointed out above. The part of the mass that had been quarried previous to the time of the author's visit is said to have expanded more uniformly and was remarkably free from strain breaks.

The suggestions following are offered with a view to improving the method of quarrying already outlined.

(1) If it is impossible to complete a line of drill holes in a corner, the corner should be cut across diagonally with a closely spaced row of holes in order that no mass too great to be crushed may remain.

(2) To avoid delay by jamming of the drills, the shear-line holes should be made first, as no appreciable movement of the rock will take place until the cross line of holes is projected.

(3) If drilling costs seem to justify it, holes should be more than 12 feet deep, as the object to be continually kept in view is to gain uniform relief at one time for as large a mass of rock as possible.

COST KEEPING.

BEARING OF COST KEEPING ON SELECTION OF METHODS AND MACHINES.

In the preceding text many different methods and types of machinery have been mentioned. Certain general advantages or disadvantages have been pointed out, but in many instances no absolute rules could be given regarding the desirability of one over another. In no two quarries are conditions alike. Variations in hardness, structure, texture, and rift, in bedding and attitude, or in unsoundness are extreme. The fact that a day's channeling for one machine varies in different quarries from 20 to 150 square feet is a typical illustration. On this account marble quarrying if conducted efficiently must involve considerable experimental investigation. A quarry operator ought to ascertain which of certain methods or machines are best adapted for his peculiar conditions. The results of past experience may enable satisfactory selection, but often experimentation must be adopted. However, no experiments convey definite information unless careful records of relative costs are kept. Therefore, cost keeping is a necessary part of such experiments. It is the yardstick that is used to measure efficiency.

COMMERCIAL RECORDS.

A certain amount of cost keeping is a recognized necessity for every successful business. Accounts that balance general expense, pay roll, fuel and supplies, maintenance and repairs, interest, and depreciation against receipts from sales, and thus furnish a basis for a statement of loss and gain on any transaction are what may be termed commercial records. Practically all quarrymen keep them in some form, though many such systems are inadequate.

TECHNICAL RECORDS.

Another and even more important phase of accounting is that dealing with detailed cost accounts for all operations in and about the quarry—accounts that are kept to supply a definite and accurate record of the relative efficiency of men, methods, and machines. Such may be termed technical records. They may be in the form of production, time-efficiency, or cost records. Cost records of each operation may be connected with commercial books by means of some suitable intermediary accounts, the nature of which should be determined by a skilled accountant.

RELATION OF TECHNICAL RECORDS TO EFFICIENCY.

Webster defines efficiency as the ratio of useful work, or of the effect produced, to the energy expended in producing it. The process of supplying the required energy costs money, and thus from

the economist's standpoint efficiency is the measure of accomplishment in terms of the cost. Increased efficiency, then, means decreased cost of production. Many books have been written on efficiency, on the relation of efficiency to organization, system, wages, etc. It is not within the scope of this bulletin to enter into all phases of business efficiency. An attempt is made simply to point out the desirability of keeping accurate and systematic records as a means of promoting efficiency. Some suggestions are offered as to the form in which such records may be kept. For those who desire to enter more fully into a study of commercial efficiency many useful books are now available.[a]

There is a tendency among many quarrymen to overestimate a loss of actual cash and underestimate a loss due to operating inefficiency, as the latter is less tangible. A mistake in bookkeeping by which a workman is paid $2 for overtime that he did not put in is considered a serious matter. If, on the other hand, by channeling a cut instead of drilling and wedging it the same man increases the cost of production of a certain amount of marble by $10, the mistake is overlooked. To the unobservant, heavy losses due to poor methods or machinery are classed as necessary burdens. Even the intelligent and observant operator may place them in the category of necessary working expenses unless a cost system reveals that they are avoidable.

The quarryman's first object should be to keep to a minimum the cost of each operation. It thus becomes his duty to continually improve his present methods, to devise more economical methods, or to maintain a low cost already realized. In order to do this, it has been stated that some efficient means must be employed for testing the various methods and determining accurately the relative value of each. A cost-keeping system supplies this need. It is a tool for cutting down costs.

Thus it may be seen that the highest efficiency demands careful technical records. It has been asserted by one economic writer that nine out of every ten failures in business are the direct result of a lack of proper knowledge of conditions, the deficiency being due to poor bookkeeping methods, or to none at all.

This is an age of competition. Manufacturers are straining every nerve to obtain contracts and to fill them at a profit. Many concerns have an uncomfortable feeling that progress and profits are not as satisfactory as they might be. Such concerns must realize that a cause must be definitely located and that the deficiency must be amended or other concerns will crowd them out. It is obvious that,

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[a] See Church, A. H., The proper distribution of expense burden, 1913, 144 pp.; Church, A. H., Production factors, 1910, 187 pp.; Gantt, H. L., Work, wages, and profits, 1913, 312 pp.; Emerson, Harrington, Efficiency, 1912, 254 pp.; Emerson, Harrington, The twelve principles of efficiency, 1912, 423 pp.; and Carpenter, C. U., Profit-making management, 1908, 146 pp.

other things being equal, the business that is scientifically managed, that can put its finger on every item of excessive cost and immediately direct energy toward a reduction of that cost, is the business that will survive when others fail.

It is thought by many that the only function of a cost system is to ascertain the cost of an article. They completely overlook the fact that a proper analysis of costs furnishes a basis for reducing costs.

QUALIFICATIONS OF SUPERINTENDENTS.

Accounts are a measure of performance but must not be interpreted as performance itself. The establishment of a good system of technical accounting does not assume increased technical efficiency. There must be (1) the ability to interpret accounts and note wherein they reflect inefficiency, and (2) the knowledge and originality necessary to devise new methods and equipment. Such qualifications must be met not only by the general manager, but by the superintendents as well. They are closely in touch with all operations and control to a large extent the methods and equipment employed. The importance of these requisites demands some elaboration.

First, the superintendent must be master of his accounts. This statement does not mean that he must be a skilled accountant. As is shown later, the accounts involved are usually of a simple nature, easily kept and easily understood. Some may feel themselves handicapped by a lack of bookkeeping knowledge. It is a safe assumption, however, that it is much easier for the technical man to attain a sufficient mastery of accounts to interpret them in terms of efficiency than for the accountant to acquire technical knowledge. A premium should therefore be placed on foremen and superintendents who have in addition to their technical training a knowledge of mathematics and bookkeeping or the ability to assimilate the necessary knowledge of these subjects. That some operators employ foremen who can not read or write is almost incredible in this age of ever-increasing commercial efficiency, but it must be accepted as a fact. It is also a fact that some such quarries are not operating at a profit.

Second, the superintendent should have wide knowledge and originality. The ability to analyze and interpret costs is of limited value without a wide knowledge of ways and means of bettering conditions. He must know of methods and equipment used elsewhere or be able on occasion to devise absolutely new ones. The previous discussion of quarry and shop operation is designed to supply necessary information on methods and equipment now in use. Few superintendents can travel widely, but they should supplement their knowledge by visiting other quarries whenever possible.

REAL PURPOSE OF TECHNICAL ACCOUNTS.

The duty of a technical accountant is not to find a convenient means of averaging and spreading costs over various operations on some arbitrary basis, but to separate and localize costs. His purpose is to enable him to follow with his cost book every operation from quarry to finished product and to see at a glance the cost of all the individual items that together constitute the total cost of production. He can then compare these costs with each other, observe which is the heaviest, and determine at what point his chief energies should be directed toward cost reduction. He can also compare the cost of the same operation through different periods of time. He can ascertain whether a new method increases or decreases the cost, and if a new machine is introduced he can determine whether it cuts down the cost of production in its limited field. Such considerations are of supreme importance. When selling prices are fixed by competition an increase in profit can be accomplished only by a decrease in cost of production. A general decrease can be achieved only by bringing about minor reductions here and there. If one is to know at once whether any new machine or method is contributing its share toward such reductions or is maintaining a minimum rate, cost keeping must be localized.

THE COST OF NOT KEEPING ACCOUNTS.

The degree of localization in cost keeping must be governed by common sense. Bookkeeping can be carried to such an extreme that the extra expense of office help will eat up the profit from increased efficiency in other departments. However, reasonably detailed systems may be employed at little extra cost. Many quarrymen are opposed to them on account of the extra office help required. One quarryman remarked, "I can not afford to install a cost system." When reminded of the successful operation of the system in one quarry he replied, ' Their margin of profit is so small that they have to keep close cost accounts." What sort of reasoning is it that leads to the conclusion that the one whose profit is presumably the larger can not afford a cost system, whereas the one whose margin of profit is narrow must burden himself with additional accountants? The two statements are in open conflict, and without doubt the second is correct. With all quarrymen the statement should be, "I can not afford to be without a cost system."

LOCALIZATION OF COSTS.

The need of cost keeping having thus been pointed out, attention may be directed toward the principles governing the localization of control accounts. In order to properly analyze costs it is necessary

to divide them into the several factors that are more or less independent of each other. There are certain factors that constitute services toward production and yet are distinct from the actual and direct processes of production. The latter are labor and supplies directly applied to the object under construction, whereas the former are such items as power, rent, interest, supervision, and yard operation. The latter are necessary factors of production, but are less directly connected with the actual processes. The purpose of the technical accountant is to keep the expense figures of these various factors separate from each other so that the efficiency of each may be judged on its own individual merits and quite independent of the influence of any other operation.

Thus, if an operator considers scrapping his steam engine and purchasing electricity at 1½ cents per kilowatt-hour, if he has no records of the present cost of his power per horsepower-hour he has no means of knowing whether the change would be profitable. If, however, he has careful records of the cost of fuel, supplies, repairs, labor, maintenance of buildings, depreciation, interest on capital, etc., for his power plant and thus is able to fix for every month the cost of power per horsepower-hour he is in a position to judge whether a change is justifiable. A special account should be kept for each of the independent factors, which together make up the total cost of production. The total of each of the indirect factors must be apportioned on a fair basis between the different units comprising the entire plant, that is, the production centers.

GENERAL AND SPECIAL ACCOUNTS.

The records kept may be classed as general and special. General accounts are of two types: The first covers operations affecting the whole plant, and the items must on this account be apportioned among the various centers of production, such as quarry, mill, and shop; the second type consists of summarized statements of the total direct and overhead expense confined to each production center independent of all others.

The method of obtaining costs for each production center is shown in the following table, where quarry, mill, and shop are represented as production centers. The total cost for each production center is the actual direct and overhead expense of each, together with the proper proportion of general expense, yard expense, and power expense for each.

Form for keeping costs of various production centers.

Quarry.		Mill.		Shop.	
Item.	Cost.	Item.	Cost.	Item.	Cost.
Direct expense..........	$........	Direct expense........	$........	Direct expense........	$........
Overhead expense.......	....	Overhead expense....	....	Overhead expense....	....
Proportion of general expense allotted to quarry	....	Proportion of general expense allotted to mill.	....	Proportion of general expense allotted to shop.	....
Proportion of yard expense allotted to quarry	....	Proportion of yard expense allotted to mill.	....	Proportion of yard expense allotted to shop.	....
Proportion of power expense allotted to quarry	....	Proportion of power expense allotted to mill.	....	Proportion of power expense allotted to shop.	....
Total, quarry......	....	Total, mill......	....	Total, shop.....	....

Other general accounts may be kept showing total production and summarized statements of total and unit costs of each operation.

In recording all general accounts it is well for purposes of comparison to place in a parallel column the cost of each item for a preceding period of possibly six months. The average figure for the period would probably be the most useful, as it would show whether an item for any given month was below or above the average cost for the preceding period.

Special accounts are localized to each special operation. Their purpose is to show the working efficiency and the time efficiency of various machines or methods. By means of careful records useful comparisons may be made. In making comparative tests of two or more machines it is wise to employ the same operator on each machine during successive periods, as the efficiency of the operators may vary more than that of the machines. This suggests that special accounts may also be kept to test the relative efficiency of men.

GENERAL ACCOUNTS AFFECTING THE WHOLE PLANT.

The general accounts [a] which must be apportioned among the various production centers are (1) general expense, (2) cost of yard operation, (3) cost of power.

GENERAL EXPENSE.

The general-expense account is made up of all items which can not be directly applied to any single production center, such as quarry, mill, or shop. It consists of the land factor and the organization factor.

[a] The writer is indebted to Maj. J. S. Sewell for many suggestions relating to quarry accounts.

LAND FACTOR.

When land is acquired for quarrying purposes either a certain amount of capital is tied up by purchase or rent must be paid. The interest or rent, together with taxes, is an annual fixed amount which must be charged against all operations conducted on the property.

ORGANIZATION FACTOR.

Costs due to organization include expenses of administration, office, advertising, and expenses of a general nature which affect the plant as a whole.

The various items of general expense are totaled at the end of each month and apportioned to the various production centers on a fixed basis, as, for example, five-eighths to quarry, one-fourth to mill, and one-eighth to shop, or on whatever other fractional basis may seem justified. A general-expense account may have some such form as the following:

Form of general-expense account.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Administration		
Office expense		
Legal expense		
Traveling expense		
Interest and discount expense		
Advertising		
Taxes		
Insurance, fire		
Insurance, liability, etc		
Total		
Charge to quarry		
Charge to mill		
Charge to shop		

YARD-OPERATION FACTOR.

The yard account includes interest on capital expended on yard equipment, wages, costs of supplies, maintenance and depreciation charges, and all other incidental expenses connected with transportation of rock from the quarry to the stock pile and from there to the mill and shop, or directly from the quarry to the mill and shop. It is a very variable account and may be omitted where the rock is loaded directly upon the transfer car with the quarry derrick. The following is a convenient form of account:

Form of account for keeping yard costs.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Superintendent		
Engineers and firemen		
Tracks:		
Labor		
Supplies		
Repairs to locomotive or crane:		
Labor		
Supplies		
Fuel		
Cars:		
Labor		
Supplies		
Miscellaneous		
Depreciation		
Total		
Charge to quarry		
Charge to mill		
Charge to shop		

POWER FACTOR.

The simplest case in power charges is where power is purchased at so much per kilowatt-hour or horsepower. The next simplest case is where the transmission and distribution is done by the purchasing company; this will involve charges for interest on capital and for depreciation and maintenance of necessary equipment in addition to the first cost. Where steam power is used, or electricity is developed by steam, there will be the interest on capital expended in boilers, engines, generators, buildings, etc., in addition to charges for maintenance, labor, fuel, and supplies. In the case of hydroelectric plants the price of fuel is usually offset by the capital invested in dams and sluices.

It is important that exact power costs be kept and wherever possible reduced to the unit cost, or rate per kilowatt-hour or horsepower-hour. A large part of the loss connected with stone quarries is in the power factor. A power cost that is excessive as compared with the numerous published statistics of power-development costs indicates that something is radically wrong and that the services of an expert are required to stop the leak. If power from more than one plant is employed a separate account should be kept for each plant in order that the expense be properly localized. The power cost should be apportioned among the various production centers on the basis of the power actually consumed by each.

When power is purchased no account is necessary except for transmission and distribution. Power plants are of various types and the form of account must be varied to suit the conditions; two typical forms are given below.

Form for costs of steam power plant.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Firemen and helpers		
Engineers		
Maintenance of buildings:		
Labor		
Supplies		
Maintenance of boilers and engines:		
Labor		
Supplies		
Maintenance of steam lines:		
Labor		
Supplies		
Fuel		
Removing cinders:		
Labor		
Supplies		
Miscellaneous		
Superintendence		
Depreciation		
Total		
Cost per unit		

In the case where steam is conducted directly to the quarry machinery the total cost may be apportioned to the various machines in the special accounts for these machines. Where part of the steam power is used to develop electricity a separate account must be kept for this item.

Cost form for hydroelectric power plant.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Attendants		
Maintenance of buildings:		
Labor		
Supplies		
Maintenance of dams and sluices:		
Labor		
Supplies		
Maintenance of power lines:		
Labor		
Supplies		
Maintenance of turbines and generators:		
Labor		
Supplies		
Supplies (oil, waste, etc.)		
Depreciation (amortization)		
Total		
Cost per kilowatt-hour		
Charge to quarry		
Charge to mill		
Charge to shop		

This account gives the proportion of power expense for each department and also indicates the efficiency of the power plant. The cost of developing electric power depends on the amount of

capital invested in dams and machinery and on the length of transmission lines and hence is subject to considerable variation. It varies generally from $1\frac{1}{2}$ to 2 cents per kilowatt-hour.

GENERAL QUARRY ACCOUNTS.

The general quarry accounts are (1) direct quarry cost, (2) overhead expense, (3) total cost of operating quarry, (4) quarry-production account, and (5) condensed cost account.

DIRECT QUARRY COST.

The direct quarry cost includes the labor and supplies required for each quarry operation. A form for keeping this account is given below:

Form for keeping direct cost accounts for quarry.

Item.	June.	
	Month.	Average for 6-month period.
Labor:	*Dollars.*	*Dollars.*
Channeling		
Setting up track		
Gadding		
Setting up gadders		
Raising blocks		
Pumping and cleaning up quarry		
Changing channeling machines		
Steel for channelers		
Steel for gadders		
Miscellaneous		
Supplies:		
Channeling		
Setting up track		
Gadding		
Setting up gadders		
Raising blocks		
Pumping and cleaning up		
Changing channeling machines		
Steel for channelers		
Steel for gadders		
Miscellaneous		
Fuel for heating		
Total direct quarry cost		

OVERHEAD EXPENSE.

The distinction between overhead and general expense is clearly defined. The latter is made up of those items which affect the plant as a whole and which must be shared by each production center proportionally. The former is a separate account kept for each production center and includes for each center indirect expenses which pertain to that center only and are independent of all others. The ordinary expenses which fall in this class are maintenance and repairs and supervision—that is, the wages of superintendents and foremen employed in this particular center of production. It seems convenient also to include depreciation charges with overhead

expenses. Depreciation is an indirect expense and is an independent item for each production center. It is usually expressed as a percentage of the original cost calculated on a basis of the estimated time which may elapse before the equipment must be replaced. It usually varies within limits of 5 to 10 per cent per annum.

Form for overhead-expense account for quarry.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Superintendent and foremen		
Watchman		
Depreciation		
Labor:		
Maintenance of channelers		
Maintenance of gadders		
Maintenance of derricks		
Maintenance of pumps		
Maintenance of buildings		
Miscellaneous		
Supplies:		
Maintenance of channelers		
Maintenance of gadders		
Maintenance of derricks		
Maintenance of pumps		
Maintenance of buildings		
Miscellaneous		
Total overhead expense for quarry		

TOTAL COST OF OPERATING QUARRY.

The total quarry-cost account contains the various items which make up the total cost of operating the quarry. It includes the proportional share of general expense, yard, and power charges, and the total direct and overhead charges for the quarry.

Form for keeping total quarry cost.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Apportioned share of general expense		
Apportioned share of yard service		
Apportioned share of power		
Total direct operating cost		
Total quarry overhead cost		
Total cost of operating quarry		

Circumstances may make it advisable to keep other accounts, such as a special development account.

QUARRY PRODUCTION ACCOUNT.

Mere statements of costs are of little value unless accompanied by a statement of production for the same period.

Form for keeping quarry production account.

Item.	June.	
	Month.	Average for 6-month period.
	Cubic feet.	*Cubic feet.*
Gross production		
Scrap blocks		
Net production		
Quantity delivered to mill (or shipped wholesale)		
Quantity delivered to yard		
Quantity delivered from yard to mill (or shipped wholesale)		

CONDENSED COST ACCOUNT.

A summarized account may be kept of the total and unit costs of each operation, including the overhead charge. It is compiled from other accounts, and presents the facts in a convenient form for reference or comparison. A convenient form for keeping such an account is given below:

Form for keeping condensed cost account.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Total cost of channeling (includes quarry overhead charge)		
Total number of square feet channeled		
Cost of channeling, per square foot		
Vertical drilling:		
Total cost (including overhead charge)		
Total number of square feet of rock drilled		
Total number of linear feet drilled		
Cost per square foot		
Cost per linear foot		
Bed drilling: [a]		
Total cost (including quarry overhead charge)		
Total number of square feet of rock drilled		
Total number of linear feet drilled		
Cost per square foot		
Cost per linear foot		
Total cost of lifting blocks (including quarry overhead charge)		
Total cost of maintaining quarry		
Total cost of power		
Total cost of quarry plant depreciation		
Total cost of development work		

[a] A term applied to drill holes made for the purpose of freeing the block from the floor.

SPECIAL QUARRY ACCOUNTS.

The accounts just considered cover the total quarry cost, the total production, and the total and unit costs of each operation. More detailed records are necessary, however, both for the purpose of compiling the general records, and to check the efficiency of each separate operation.

In order to test channeling or drilling efficiency no account is taken of the proportion of waste, for it makes no difference whether sound or unsound material is produced, as regards the rate of cutting. Hence all such records are based on gross production. For channeling the following records should be kept:

Form for keeping channeling costs, based on gross production.

Item.	Month.
Square feet channeled....................................	....
Channeling cost..	....
Square feet cut in 10-hour day...........................	....
Square feet cut in 10-hours' cutting time................	....
Channeling cost per square foot..........................	....
Square feet channeled per cubic foot.....................	....
Channeling cost per cubic foot...........................	....

Form for channeling time-efficiency account.

Time.	Month.
Maximum working hours....................................	....
Cutting time, per cent...................................	....
Time used for repairs, per cent..........................	....
Time used in moving and setting track, per cent..........	....

The following gadding accounts should be kept:

Account for keeping gadding costs, based on gross production.

Item.	Month.
Square feet of gadding...................................	....
Gadding cost...	....
Square feet cut in 10-hour day...........................	....
Square feet cut in 10 hours' cutting time................	....
Gadding cost per square foot.............................	....
Square feet of gadding per cubic foot....................	....
Gadding cost per cubic foot..............................	....
Linear feet cut..	....
Cost per linear foot.....................................	....

Account for keeping gadding time efficiency.

Time.	Month.
Maximum working hours....................................	....
Cutting time, per cent...................................	....
Time used for repairs, per cent..........................	....
Time used for moving, per cent...........................	....

Separate accounts may be kept for vertical drilling and bed drilling, or they may be united. Both channeling and gadding accounts may for purposes of comparison be localized to machines of the same class or even to individual machines.

Such accounts also afford a means of testing various methods. Suppose a change is made in the direction of a row of drill holes so that advantage is taken of a rift in the rock which permits a wider spacing of the holes. The result would be shown in the decreased gadding cost per cubic foot of rock produced. The exact saving thus effected may be balanced against any increased cost which may be involved in the change of quarry method, and thus the operator will know whether such a plan is justified.

As a further illustration, suppose the rift to be so poorly defined that it may seem advisable to channel in two directions and gad only on the bed. Such a change would lower the gadding cost and increase the channeling cost per cubic foot of rock produced. The sum of the channeling and gadding costs before the change compared with their sum after the change would indicate the loss or gain involved in such a change of method.

The preceding statements indicate that in many cases conclusions must not be drawn from isolated accounts, but from the relation of one account to another. The accounts are often interdependent, and no one can interpret them properly except the man who is familiar with both the quarry operations and the accounts kept.

A very useful account is one which summarizes the unit cost of each item on a basis of gross production followed by the same item on a basis of net production, as follows:

Account for keeping gross and net unit costs.

Item.	June.			
	Month.		Average for 6-month period.	
	Gross.	Net.	Gross.	Net.
	Dollars.	*Dollars.*	*Dollars.*	*Dollars.*
Channeling, per cubic foot	....	....	....	....
Vertical drilling, per cubic foot	....	....	....	....
Bed drilling, per cubic foot	....	....	....	....
Lifting blocks, per cubic foot	....	....	....	....
Maintenance of quarry, per cubic foot produced	....	....	....	....
Power, per cubic foot produced	....	....	....	....
Total cost	....	....	....	....

Such an account indicates clearly how the proportion of waste adds to the cost of production of the finished product.

For a certain month the total cost of production per cubic foot, gross measure, in one marble quarry, was only 35 cents. When calculated

on the basis of net production it was, however, $1.31 per cubic foot. This means that if all the marble produced were salable—an ideal condition which is impossible in practice but may be assumed for purposes of illustration—the marble could be sold at 36 cents a cubic foot for a profit, if selling expenses were disregarded. So much rock is thrown away on the waste heap, however, that with no compensating revenue from waste material the good stock must be sold for more than $1.31 in order to realize a profit. If more quarrymen kept records which would clearly indicate the excessive losses that result from quarrying inferior material, it is safe to assume that more active steps would be taken in an attempt to modify quarry methods in such a manner as to reduce the proportion of waste.

Block records are very useful, as they assist the quarry operator in keeping track of his product. A form of record kept by one marble company is as follows:

Block record kept by a North Carolina company.

No. of block.	Date quarried.	Dimensions.	Grade.	Total cubic feet.	To mill.
1243.........	*July 7, 1914*	*3′ 0″ by 3′ 0″ by 1′ 0″.*	*No. 3*	*9*	*August 4, 1914*
....	....	....	....	....	....
....	....	....	....	....	....
....	....	....	....	....	....

For companies who sell their stock wholesale a block record is still more desirable and requires greater detail. A form used by a Tennessee marble company, which is very complete and enables the operator to trace every block from the quarry pit to its final destination, is given below:

Form of block record kept by wholesale company.

Date listed.	List No.	Grade.	Color.	Official measure.	Cubic feet.	Accepted measure.
May 10	*1234*	*1*	*Pink*	*7′ 0″ by 6′ 0″ by 3′ 0″.*	*126*	....
May 10	*1235*	*1*	...*do*...	*6′ 6″ by 6′ 0″ by 3′ 0″.*	*117*	....
....	....	....	....	....	....	....
....	....	....	....	....	....	....
....	....	....	....	....	....	....
....	....	....	....	....	....	....

Form of block record kept by wholesale company—Continued.

Cubic feet.	Description and condition.	Consignee.	Address.	Charged.	Address.
....	*Good block, regular shape, fine color.*	....	....	....	....
....	....*do*..........	....	....	....	....
....	....	....	....	....	....
....	....	....	....	....	....
....	....	....	....	....	....
....	....	....	....	....	....

Order No.	Date shipped.	Folio.	Remarks; car No., routing, etc.
18256........	*June 14*....	*152*	*Southern 1181189, c/o Mobile and Ohio.*
18257........	.....*do*.....	*152*	*Do.*
....	....	....	....
....	....	....	....
....	....	....	....
....	....	....	....

The company intends to supplement this report with a sketch of each block showing the position of cutters if the block is unsound, and the position of any irregularities or flaws. Such sketches would be placed on file for reference in case of a dispute with the buyer.

GENERAL MILL ACCOUNTS.

The general mill accounts are (1) direct mill cost, (2) overhead expense, (3) total cost of operating mill, (4) unit cost account, (5) condensed cost account.

DIRECT MILL COST.

The direct mill cost includes charges for labor and supplies for the various processes in the mill. A form for keeping this account is given below.

Form for keeping direct mill costs.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Labor:		
Loading and unloading gangs		
Sawing		
Hoisting engineer		
Placing stock		
Sawing with diamond saw		
Removing yard scrap and waste		
Loading sawed stock in cars		
Bracing sawed stock in cars		
Water supply		
Supplies:		
For loading and unloading gangs		
For hoisting engineer		
For placing stock		
For removing yard scrap and waste		
For loading sawed stock in cars		
For bracing sawed stock in cars		
Saw blades		
Sand		
Total direct mill cost		

OVERHEAD EXPENSE.

Overhead expense includes cost of superintendence, maintenance and repairs, and depreciation. A form for keeping an overhead-expense account for the mill is given below.

Form of overhead-expense account for mill.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Superintendent		
Foreman		
Depreciation		
Labor:		
Maintenance of gangs and sand pumps		
Maintenance of diamond saws		
Maintenance of crane and runway		
Maintenance of shafting and belting		
Maintenance of drain and settling basin		
Maintenance of buildings		
Maintenance of machine shop and equipment		
Maintenance of heating		
Watchman		
Miscellaneous		
Supplies:		
Maintenance of gangs and sand pumps		
Maintenance of diamond saws		
Maintenance of crane and runway		
Maintenance of shafting and belting		
Maintenance of drain and settling basin		
Maintenance of buildings		
Maintenance of machine shop and equipment		
For heating		
For watchman		
Miscellaneous		
Total overhead		

TOTAL COST OF OPERATING MILL.

A form for keeping a summarized statement of mill costs, similar to that for the quarry, is given below:

Form for keeping total costs of operating mill.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Apportioned share of general expense		
Apportioned share of yard service		
Apportioned share of power		
Total direct operating cost		
Total overhead charge for mill		
Total cost of operating mill		

UNIT COST ACCOUNT.

The unit cost account shows the amount of material produced and the unit costs direct and indirect for gangs only. If diamond saws or wire saws are employed, separate accounts should be kept for them.

Form of unit cost account for gangs.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Number of gang hours actually made		
Direct cost per gang hour		
Total cost per gang hour		
Square feet of saw cuts		
Direct cost per square foot of saw cut		
Total cost per square foot of saw cut		
Cubic feet of blocks for first cut		
Cubic feet of blocks for second and third cuts		
Cubic feet of blocks for all stock loaded		
Total cubic feet sawed (all between saws)		
Total cubic feet good color obtained		
Total cubic feet good and sound		
Direct cost per cubic foot total sawed		
Total cost per cubic foot total sawed		
Direct cost per cubic foot good color		
Total cost per cubic foot good color		
Direct cost per cubic foot good and sound		
Total cost per cubic foot good and sound		
Average square feet saw cuts per gang hour		
Average square feet sawed per gang hour		
Rate of sawing		
Average number of blades per gang load		
Average length of stock		
Average number of cubic feet per gang load		
Number of gang loads		

This form of account is subject to many modifications. The one given is designed to fit a complex condition, where the marble is marred by both bad color and unsoundness. If either of these is absent the account may be simplified. If the material is graded as No. 1, 2, and 3 as it leaves the saws the account must be modified to suit.

CONDENSED COST ACCOUNT.

The value of material as it enters the mill is the sum of the costs of work already done upon it as indicated by the quarry records. The summarized account gives in condensed form the cost of material through the mill and indicates the additional costs of various operations, both as totals and units.

Form for keeping condensed cost account.

Item.	June.	
	Month.	Average for 6-month period.
	Dollars.	*Dollars.*
Value of blocks to mill from quarry		
Value of blocks to mill from yard		
(*a*) Total value of blocks to mill		
Average value per cubic foot of blocks to mill		
(*b*) Total mill operating cost		
Total value of output of mill (sum of *a* and *b*)		
Thin stock: [a]		
Value of blocks delivered to mill		
Total cost of sawing		
Total value of sawed stock produced		
Average value of sawed stock produced		

[a] Similar accounts may be kept to ascertain the costs per cubic foot of making the first, second, and third cuts, cuts subsequent to the third, and of cutting with the diamond saw.

For convenience in keeping saw-gang records, the accompanying ticket form which is used by one marble company may offer useful suggestions:

SAW TICKET.

Gang No. .... *Dimensions* .... *Block No.* .... *Grade* ....

	Date.	Hour.	
When last down			No. saws.....
When started			No. pcs.......
When down			

Total hours under gang ...... *Idle for loading* ...... *Sizes sawed into* ......

HOURS RUN AND INCHES SAWED EACH SHIFT.

Sawyer.	1		3		5		7	
	Hrs.	Ins.	Hrs.	Ins.	Hrs.	Ins.	Hrs.	Ins.
	2		4		6		8	
Totals								

Total hours sawed ...... *Total inches sawed* ...... *Total hours delay* ......

DELAYS.

Date.	Hours.	Cause.
....	....	....
....	....	....
....	....	....
....	....	....

SPECIAL MILL ACCOUNTS.

The special mill accounts show the amounts produced by the various operations and the total and unit costs of each operation, with certain other useful figures such as rate of sawing, number of blades, etc. A summarized statement of total quantities produced and cost of the same may be given as follows:

Form for keeping cost account of total quantities produced.

Total quantities produced by gang.	June.	
	Month.	Average for 6-month period.
Gang hours actually made	....	....
Direct cost per gang hour	....	....
Total cost per gang hour	....	....
Square feet of saw cuts	....	....
Direct cost per square foot of saw cut	....	....
Total cost per square foot of saw cut	....	....
Gross measure blocks for first cut	....	....
Gross measure blocks for second and third cuts	....	....
Gross measure all stock loaded	....	....
Total [a] cubic feet sawed (all between saws)	....	....
Direct cost per cubic foot (total sawed)	....	....
Total cost per cubic foot (total sawed)	....	....
Average number of square feet sawed per gang hour	....	....
Average number of cubic feet sawed per gang hour	....	....
Rate of sawing	....	....
Average number of blades per gang load	....	....
Average length of stock	....	....
Average number of cubic feet per gang load	....	....
Number of gang loads	....	....

[a] Similar items showing the cost of producing the different grades may be given if desired.

A thin-stock account may be kept in more or less detail. The form given here may be modified at the operator's discretion.

Form for keeping thin-stock account.

Thin stock.	June.	
	Month.	Average for 6-month period.
Gross measure, cubic feet		
Total [a] cubic feet sawed (all between saws)		
Total number of square feet of saw cuts		
Total gang hours		
Direct operating cost		
Total operating cost		
Direct cost per cubic foot of total quantity sawed		
Total [a] cost per cubic foot of total quantity sawed		
Direct cost per square foot of saw cut		
Total cost per square foot of saw cut		
Average number of square feet sawed per gang hour		
Average number of cubic feet cut per gang hour		
Average rate of sawing		
Average number of blades per gang load		
Average length of stock		
Average number of cubic feet per gang load		
Total number of gang loads		

[a] Items similar to this one for the different grades might be inserted here.

Similar accounts may be kept for cubic stock, showing the cost of making the first cut, second cut, third cut, and cuts subsequent to the third cut, and for diamond saws. Cuts subsequent to the third cut are usually diagonal or some such direction for the purpose of decreasing mill labor, and as a consequence are charged to the mill.

Time-efficiency reports are of great value. The various causes of delay may be enumerated and calculated to percentages of the maximum sawing hours, as in the form following:

Form for time-efficiency record of gang.

Item.	June.			
	Month.		Average for 6-month period.	
	Hours.	per cent.	Hours.	per cent.
Maximum sawing time				
Actual sawing time				
Time consumed by delays:				
Loading and unloading				
Coming down at night				
Without water				
Without power				
Repairs to gangs				
Repairs to belting and shafting				
Repairs to pumps				
Repairs to lead pipes				
Without blocks				

Suppose that in addition to a traveling crane a transfer-car system is installed. In the next monthly report the advantage of this additional equipment would be reflected in the time-efficiency record as a material reduction in the time consumed in loading and unloading.

SHOP OR FINISHING PLANT ACCOUNTS.

In considering the various operations in the shop a new condition presents itself. Although the whole shop has been termed a production center it is in reality a group of smaller independent production centers comprising coping, rubbing plain stock, gritting plain stock, buffing plain stock, cutting, matching, drilling holes for anchors, boxing, and loading. Keeping in mind the principle that technical accounts should be localized if costs are to be properly isolated, for each of these smaller production centers, a separate account must be kept, each bearing its proper share of general expense, yard service and power, and its total direct and overhead charge. The apportioning of general expense and other indirect costs requires careful adjustment. Some departments require more supervision or more yard service than others, and each should be burdened with no more than its fair share. Power cost is apportioned on the basis of the actual amount of power used by each department.

Two general shop accounts are necessary, one being a summary of shop costs and the other a summary of shop production.

Cost account for shop or finishing plant.

Item.	June.	
	Month.	Average for 6-month period.
General expense:	*Dollars.*	*Dollars.*
Percentage of main general expense account		
Draftsmen (percentage of total)		
Drafting supplies (percentage of total)		
Total general expenses		
Yard operation (percentage of total)		
Power (amount actually used)		
Direct shop costs:		
Labor		
Supplies		
Total direct shop costs		
Overhead expense for shop:		
Draftsmen (percentage of total)		
Drafting supplies (percentage of total)		
Superintendence		
Shop depreciation		
Maintenance of shop machinery—		
Labor		
Supplies		
Total overhead expense for shop		
Summary:		
General expenses		
Yard operation		
Power		
Direct shop costs		
Overhead expense for shop		
Total cost of operating shop		
Total number of productive hours of shop labor		
General expense per productive hour		
Yard expense per productive hour		
Power expense per productive hour		
Direct expense per productive hour		
Overhead expense per productive hour		

A plain-stock production account summarizes the number of cubic feet of stock handled and the work done on it at each operation.

Form for keeping records of plain-stock production.

Plain-stock production.	June.	
	Month.	Average for 6-month period.
Square feet of stock coped (excluding tile)		
Cubic feet of stock coped (excluding tile)		
Square feet of tile coped		
Cubic feet of tile coped		
Square feet of stock rubbed (excluding tile)		
Square feet of second surface rubbed (excluding tile)		
Total square feet rubbed		
Cubic feet of stock rubbed		
Square feet of tile rubbed		
Cubic feet of tile rubbed		
Square feet of stock gritted		
Square feet of second surface gritted		
Total square feet gritted		
Cubic feet of stock gritted		
Square feet of stock buffed		
Square feet of second surface buffed		
Total square feet buffed		
Cubic feet of stock buffed		

The account for each of the production centers within the shop may have somewhat the following form:

Form for keeping costs of individual operations.

Coping.	June.	
	Month.	Average for 6-month period.
Total number of productive hours		
Square feet of stock coped		
Cubic feet of stock coped		
General expense		
Yard service		
Power		
Overhead shop expense		
Direct expense:		
Copers		
Helpers		
Foremen		
Removing scrap		
Supplies		
Total cost of coping per cubic foot (*a*)		
Cost of stock per cubic foot before coping (*b*)		
Value of coped stock per cubic foot (sum of *a* and *b*)		

In this account the value of the stock is the sum total of the costs of all previous operations. After it has been coped the stock will have increased in value by the amount of the coping costs. The same process is followed during all subsequent operations, and the final boxing and loading account will show as the value of the stock the total cost of all operations through quarry, mill, and shop. The

accounts for rubbing plain stock, gritting plain stock, buffing plain stock, cutting, matching, and other necessary processes are kept in the same form as the coping account, though certain modifications must be made in the direct expense of each. Tile stock or refinishing may require additional accounts.

Accounts should be kept for all by-products, such as terrazzo, road material, fertilizer, riprap, lime, etc.

JOB ACCOUNTS.

In shops where novelties are manufactured it may seem desirable to determine the total cost of an article which has passed through the hands of a number of operators. Such a record when placed on file affords a ready means of making future estimates on similar articles. If a block of cubic stock is taken, its first cost may easily be ascertained from the total cost of third-cut cubic stock in the condensed cost account of the mill records. The final cost will be this amount plus the cost of all subsequent work done upon it. For keeping such records the following forms of cost tickets are suggested. When the job is completed, the information contained in ticket 1 is transferred in condensed form to ticket 2. The latter may be of cardboard, of convenient size and shape for filing.

Suggested forms for cost tickets.

COST TICKET NO. 1.

Order No. .... Design..............................

Kind of marble........................ Size ....

Article..............................

Class of workman.	Name of workman.	Time started.	Time stopped.	Total time.	Rate.	Cost.	Remarks.
Cutter............	*L. Smith*..	*7.30 a. m. 6/27/14*	*11 a. m. 6/27/14*	*3½ hrs.*	*50¢*	*$1.75*	
Cutter............	....	....	....	....	....	....	
Cutter............	....	....	....	....	....	....	
Polisher..........	....	....	....	....	....	....	
Polisher..........	....	....	....	....	....	....	
Tracer............	....	....	....	....	....	....	
Turner............	....	....	....	....	....	....	
Lathe polisher....	....	....	....	....	....	....	

COST TICKET NO. 2.

Design........... Order No. .... For............ Price............

Sizes.	Cubic feet.	Cost of—					Total cost.
		Cutting.	Polishing.	Tracing.	Turning.	Lettering.	
....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....

PUBLICATIONS ON MINERAL TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines is temporarily available for free distribution. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

BULLETIN 16.—The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 42.—The sampling and examination of mine gases and natural gas, by G. A. Burrell and F. M. Seibert. 1913. 116 pp., 2 pls., 23 figs.

BULLETIN 45.—Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 47.—Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp.

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BULLETIN 81.—The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs.

BULLETIN 84.—Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

TECHNICAL PAPER 3.—Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp.

TECHNICAL PAPER 8.—Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 14.—Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

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TECHNICAL PAPER 38.—Wastes in the production and utilization of natural gas, and means for their prevention, by Ralph Arnold and F. G. Clapp. 1913. 29 pp.

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TECHNICAL PAPER 43.—The influence of inert gases on inflammable gaseous mixtures, by J. K. Clement. 1913. 24 pp., 1 pl., 8 figs.

TECHNICAL PAPER 50.—Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

TECHNICAL PAPER 66.—Mud-laden fluid applied to well drilling, by J. A. Pollard and A. G. Heggem. 1914. 21 pp., 12 figs.

TECHNICAL PAPER 68.—Drilling wells in Oklahoma by the mud-laden fluid method, by A. G. Heggem and J. A. Pollard. 1914. 27 pp., 5 figs.

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